



THE
ONTARIO WATER RESOURCES
COMMISSION

INDUSTRIAL WASTES SURVEY

of

STANROCK URANIUM MINES LIMITED

ELLIOT LAKE, ONTARIO

TD
428
.U73
S87
1969

copy 2
MOE

1969

Copyright Provisions and Restrictions on Copying:

This Ontario Ministry of the Environment work is protected by Crown copyright (unless otherwise indicated), which is held by the Queen's Printer for Ontario. It may be reproduced for non-commercial purposes if credit is given and Crown copyright is acknowledged.

It may not be reproduced, in all or in part, for any commercial purpose except under a licence from the Queen's Printer for Ontario.

For information on reproducing Government of Ontario works, please contact ServiceOntario Publications at copyright@ontario.ca

TD
428
.U73
S87
1969
copy 2

A supplementary report on an
industrial wastes survey of
Stanrock uranium mines limited,
Elliot Lake, Ontario.

80470

A

Supplementary Report On

AN INDUSTRIAL WASTES SURVEY

of

STANROCK URANIUM MINES LIMITED

Elliot Lake, Ontario

1969

Division of Industrial Wastes

ONTARIO WATER RESOURCES COMMISSION

TABLE OF CONTENTS

	<u>Page No.</u>
SUMMARY	1
DETAILS OF SURVEY	2
Personnel Participating	2
HISTORICAL SUMMARY	3
GEOLOGY-MINERALOGY	5
MINING (1958 - 1964)	13
Mine Development	13
Services and Equipment	
(i) Ventilation	16
(ii) Compressed Air	16
(iii) Drill Water	16
(iv) Pumping and Drainage	17
(v) Shaft Construction	17
(vi) Drilling and Blasting	18
(vii) Bacterial Leaching (1960 - 1970)	19
MILLING	
(1) Past Milling Procedures	22
Original Flow Sheet	23-29
(ii) Present Milling Procedures (1969)	30
Present Flow Sheet	31
PRODUCTION DATA	32
COMPANY PROPERTIES	33

Ore Reserves	34
SALES CONTRACTS	34
WATER SUPPLY	
(a) Domestic Supply	35
(b) Industrial Supply	36
TAILINGS DISPOSAL - CHRONOLOGICAL HISTORY	37
Tailings Area - Mining Claim Boundaries	39
SOURCES OF LIQUID WASTES	
(a) Mine-Mill Complex	39
(b) Townsite	41
(c) Tailings Area	41
(A) Decant Overflow	43
(B) Decant Swamp Overflow	45
(C) South Dam Seepage	47
(D) Mine Road Seepage	48
(E) Orient Creek Swamp	49
(F) North Dam Seepage	50
(G) Swallow Hill Seepage	52
(H) Seepage to Beaver Lake	54
(I) Beaver Creek	55
Lakes and Watercourses Influenced Directly by the Stanrock Tailings Area	
(1) Cinnabar Creek	56

Page No.

(2) Orient Creek	58
(3) Beaver Creek	58
(4) Beaver Lake	59
(5) Pink Dragon Lake	62
(6) Half Moon Lake	67

SOURCES OF SOLID WASTES

(a) Conventional Mining and Milling Operations	68
(b) Bacterial Leaching	69

CHARACTERISTICS OF THE STANROCK TAILINGS AREA

(a) Size	70
(b) Depth of Tailings	71
(c) Precipitation	71

STABILITY OF THE STANROCK TAILINGS AREA

(a) North Dam	71
(b) Swallow Hill Area	72
(c) South Dam	73

DEFLATION (WIND EROSION) ON THE STANROCK TAILINGS AREA	77
--	----

SAMPLING AND ANALYSIS

(a) Weather Conditions During Sampling Periods	83
(b) Sampling	83
(c) Analysis	83

WASTE LOADINGS (GROSS)

Part A - Waste Loadings to Quirke Lake

(a) North Dam Seepage	84
(b) Swallow Hill Seepage	85
(c) Beaver Creek	86

Total Gross Loadings to Quirke Lake	86
-------------------------------------	----

Part B - Waste Loadings to the Lower Serpent River System

(a) Decant Overflow	86
(b) South Dam Seepage	87
(c) Mine Road Seepage	88
(d) Orient Creek Swamp Runoff	88

The Total Gross Loadings <u>from the immediate tailings area</u> to the Lower Serpent River System	88
---	----

Waste Loadings - Pink Dragon Lake Overflow	89
--	----

TOTAL WASTE LOADINGS TO THE SERPENT RIVER WATERSHED FROM STANROCK URANIUM MINES LIMITED	89
--	----

RADIOLOGICAL DATA

(a) Quirke Lake Intake Water	90
(b) Mine Water to Mill	90
(c) Barren to Mine	91
(d) North Dam Seepage	91
(e) Tailings Area Decant	91

(v)

Page No.

(f) Influent to "Spill" Area	92
(g) Effluent from "Spill" Area	92
(h) Beaver Lake Overflow	92
(i) Combined North Dam Seepage-Swallow Hill Seepage	93
(j) Beaver Creek at Quirke Lake	93

"MPC (Maximum Permissible Concentration) Objectives for Drinking Water Contaminated by Certain Uranium Thorium Daughter Mixtures"	93-103
---	--------

CHRONOLOGICAL HISTORY OF WASTE TREATMENT AT STANROCK URANIUM MINES LIMITED	105
--	-----

THE BEGINNING OF POSITIVE WASTE TREATMENT AT STANROCK URANIUM MINES LIMITED	112
---	-----

COMMENTS CONCERNING THE DIRECTION OF FLOW OF WASTES FROM THE VICINITY OF STANROCK URANIUM MINES LIMITED AND THE DECISIONS INVOLVED	119
--	-----

(i) The First Decision	120
------------------------	-----

(ii) The Second Decision	122
--------------------------	-----

COMMENTS - THE IMPLEMENTATION OF THE FIRST AND SECOND DECISIONS	122
---	-----

ADDITIONAL WASTE TREATMENT FACILITIES REQUIRED	124
--	-----

GENERAL CONCLUSIONS	126
---------------------	-----

RECOMMENDATIONS	126
-----------------	-----

APPENDIX A

CAN-MET EXPLORATIONS LIMITED

General

Page No.

Geology	2
Milling	3
Production	3

APPENDIX B

WATER QUALITY OBJECTIVES - POLICY GUIDELINES	1-3
OBJECTIVES FOR WATER QUALITY CONTROL IN THE PROVINCE OF ONTARIO	4-8
OBJECTIVES FOR INDUSTRIAL WASTES CONTROL IN ONTARIO	9-10

APPENDIX C

"BACKGROUND" DATA (Elliot Lake, Ontario)	1
--	---

APPENDIX D

TAILINGS AREA - STRUCTURAL DETAIL	1-7
-----------------------------------	-----

APPENDIX E

IRON (FERROUS-FERRIC) IN THE STANROCK EFFLUENTS	
(a) Source	1
(b) Reaction Chemistry	
(i) Sulphide to Sulphate	2
(ii) Oxidation of Iron (Fe^{+2} to Fe^{+3})	2
(iii) Precipitation of Iron	3

APPENDIX F

MICROBIOLOGICAL LEACHING OF SULPHIDE MINERALS	1-4
---	-----

The following report should be regarded as a supplementary report concerning Stanrock Uranium Mines Limited. The original report entitled "Industrial Wastes Survey of Stanrock Uranium Mines Limited - Elliot Lake, Ontario - 1968" served to delineate the specific problems facing the Company and, at the same time, attempted to suggest causes and cures for the various situations as they arose.

During the 1969 field season, the Division of Industrial Wastes obtained a detailed set of "environmental" water samples in and around the Stanrock property, and it is these samples and the field observations associated with these samples that form the basis of this report.

SUMMARY

Stanrock Uranium Mines Limited is solely responsible for the present serious impairment of Beaver Lake, Pink Dragon Lake and Half Moon Lake. The Company is partially responsible for the impairment of Quirke Lake, May Lake, Hough Lake and Pecors Lake.

The present daily waste loadings from the Stanrock operation to the Serpent River watershed are as follows:

Total Iron as Fe	-	4,015 lbs.
Ferrous Iron	-	1,130 lbs.
Sulphate as SO ₄	-	19,070 lbs.
Uranium as U	-	15 lbs.
Zinc as Zn	-	33 lbs.
Nickel as Ni	-	10 lbs.

Cobalt as Co	-	7 lbs.
Copper as Cu	-	10 lbs.
Manganese as Mn	-	30 lbs.

The above list is incomplete and can easily be extended to include several other "heavy" metals.

DETAILS OF SURVEY

Stanrock Uranium Mines Limited was visited during the months of June and October in 1969. Normal mill samples were obtained along with fairly detailed sets of samples that were intended to demonstrate the effect of this uranium operation on its surroundings.

The conclusions drawn in this final report were based on the facts available. Assumptions, when used, are clearly defined.

Personnel Participating

Mr. J. R. Hawley visited the Stanrock operation several times during the months of June and October. On many of these occasions he was accompanied by highly qualified OWRC personnel from the Division of Industrial Wastes and from the Division of Laboratories. During the month of October 1969, for instance, Mr. J. N. Bishop (Chemist-in-Charge, Chemistry II) travelled to the Stanrock operation in order to carry out on-the-spot determinations of ferric and ferrous iron. At the same time, Mr. Bishop investigated methods of sampling and sample preservation in order that the analytical results presented in this and subsequent reports be as accurate as possible.

Many unusual solid and semi-solid samples were taken in and around the Stanrock tailings body. The identification of these samples and the associated detailed technical work was skillfully carried out by Mr. D. Robinson, also working with Chemistry II in the Division of Laboratories.

Special technical assistance was received from the Division of Research.

Personnel interviewed at Stanrock Uranium Mines Limited included Mr. B. G. MacDermid (Mine Manager), Mr. R. D. MacGregor (Mine Superintendent) and Mr. J. Williamson (Mill Foreman).

HISTORICAL SUMMARY

The Stanrock Mine is situated on the east shore of the west arm of Quirke Lake, 15 miles by road from the Town of Elliot Lake.

The property, originally known as the "Z-7 group", was purchased by Stancan Uranium Mines Limited in 1954 from Zenmac Metal Mines Limited.

In 1955 and 1956, Stancan carried out a diamond-drilling programme that indicated a probable commercial uranium deposit on the Z-7 group of claims (19 claims and 3 fractions; 615 acres, Townships 144 and 150, Quirke Lake sector, Elliot Lake, Ontario). Seven drill holes cut ore-grade material.

In 1956, Stancan, an American Corporation, formed a Canadian subsidiary known as Stanrock Uranium Mines Limited to operate the property.

Stanrock began construction work on a mining plant in 1956. During 1956-1957, two shafts, approximately 800 feet apart, were sunk on the property. The first, a three-compartment production shaft, was carried to a depth of 3,379 feet

while the second, a two-compartment service shaft, bottomed at 2,952 feet. The shafts were connected underground.

The original target date for mill start-up, November 1957, was not met. Production commenced March 1, 1958 at an initial rate of 1,000 tons per day. By the year's end, production had increased to 3,000 tons per day (capacity rate).

The late start-up and a substantial overrun on construction costs together made it impossible for the Company to meet the financial schedule. Further difficulties were encountered when the ore beds were found to average only 6 feet in thickness instead of the drill-indicated thickness of 11 to 12 feet. Early in 1959, the mining programme had to be revised.

Stanrock entered voluntary receivership in May 1959, and the Montreal Trust Company was appointed receiver and manager. In any case, operations have continued on an uninterrupted basis, and it is expected that the original contract (delivery of 9,198,000 pounds of U_3O_8) for production of uranium concentrates will be completed.

By early 1964, the Company had discharged its remaining obligations under the bankruptcy proceedings and by January 1965 it was, for the first time since 1956, free of all long-term debt.

As high grade ore reserves were depleted, mining emphasis gradually shifted (beginning in 1960) from conventional underground practice to uranium extraction by bacterial leaching.

Conventional underground mining ceased at the end of October in 1964.

At that time, the Company expected to continue shipment of concentrates until the end of 1965. The recovery of U_3O_8 from the property by bacterial leaching was expected to be approximately 15,000 pounds per month. Prior to conversion to this method, the monthly output was about 150,000 pounds. The small amount of uranium oxide that remained early in 1965 to be delivered under contract was to be "mined" by bacterial leaching and required only a skeleton staff.

GEOLOGY-MINERALOGY

The Stanrock property and mine were well suited for bacterial leaching. The Stanrock orebodies are in a fringe area that is downplunge from the uraniferous quartz-pebble conglomerate beds on the properties of Denison Mines Limited and the Spanish American Division of the Rio Algom group.

The orebody itself was part of an extensive flat-lying reef with an average dip of 14 to 15 degrees to the southwest. As a result, mine waters were able to flow slowly for considerable distances through worked-out areas **picking** up additional uranium values before being collected in sumps and pumped to the surface. The west (abandoned) section of the mine in which leaching originally took place had been mined down dip to the southwest corner of the property, thereby providing practically unlimited natural sump capacity. In addition, Stanrock's mine drainage system had been designed to handle 2 to 3 times the volume of water (most of the ore underlies Quirke Lake) actually encountered in mining operations.

The prime prerequisite for the type of leaching to be carried out was a sufficient supply of iron and sulphur - this supply coming preferably from the ore

itself. The Stanrock ore filled the requirements.

All the orebodies in the Elliot Lake district (including the Stanrock orebody) are contained in a quartz-pebble conglomerate in the lower part of the Mississagi formation, the basal formation of the Bruce series. Some of the beds are low grade and/or too thin to be mined. The mineable bodies appear to lie in pronounced depressions on the pre-Huronian surface. The thickest beds of conglomerate occur in the lowest parts of the depressions, and are underlain by green schists in contrast to pre-Huronian granitic rocks that underlie the thin parts of the conglomerate. The main ore-bearing conglomerate is not at the base of the Mississagi, and at the Quirke and Nordic mines at least, in addition to a continuous bed of ore grade, there are other conglomerate beds above and below - all in the Mississagi.

The uranium and rare-earth values in the ore are concentrated (as microscopic mineral grains) in the matrix of the quartz-pebble conglomerate. The matrix itself is composed essentially of **fine** quartz (SiO_2) grains, sericite $((\text{H},\text{K})\text{AlSiO}_4)$, and a small proportion of chlorite $(\text{H}_4\text{Mg}_3\text{Si}_2\text{O}_9 > \text{H}_4\text{Mg}_2\text{Al}_2\text{SiO}_9)$. Important gangue minerals of the Stanrock ore are pyrite (FeS_2) and pyrrhotite (Fe_7S_8) both of which occur as constituents of the matrix.

The radioactive ore minerals named in order of abundance are generally brannerite, uraninite, and monazite. One mine in the area reports that brannerite, uraninite, and monazite constitute 60 per cent, 30 per cent, and 10 per cent respectively of the uranium content of the ore.

The following analyses of the principal ore minerals are intended to be a guide to the provenance of the various contaminants found in the waste discharges from the operation:

MINERAL: brannerite

Formula: $((U,Ca,Fe,Y,Th)_3Ti_5O_{16})$

The mineral is essentially an oxide of titanium, uranium, and calcium, with minor yttrium, thorium, and ferrous iron.

Typical Analysis:

<u>Component</u>	<u>Per Cent</u>
CaO	2.9
BaO	0.3
SrO	0.1
PbO	0.2
FeO	2.9
$(Y,Er)_2O_3$	3.9
UO ₂	10.3
UO ₃	33.5
ThO ₂	4.1
ZrO ₂	0.2
SiO ₂	0.6
TiO ₂	39.0
CO ₂	0.2

H₂O 2.0

Total 100.2

NOTE: The state of oxidation of the Fe and U is uncertain. The water content is probably due to alteration. Contains Helium.

MINERAL: uraninite

Formula: UO₂

NOTE: The natural material is always more or less oxidized, and the actual composition lies between UO₂ and U₃O₈ (U₃₋₄O₈) with U⁴ usually predominant.

Typical Analysis:

<u>Component</u>	<u>Per Cent</u>		
	<u>1</u>	<u>2</u>	<u>3</u>
CaO	1.01	0.35	1.72
MnO	0.03	0.14	0.13
PbO	10.95	11.69	16.71
MgO	0.08	0.06	0.06
(Y,Er) ₂ O ₃	2.14	0.73	1.19
(Ce,La) ₂ O ₃	1.88	0.06	0.28
La ₂ O ₃	--	--	--
CeO ₂	--	--	--
UO ₂	39.10	--	--
U ₃ O ₈	--	86.16	64.86
UO ₃	32.40	--	--
ThO ₂	10.60	0.10	13.94

<u>Component</u>	<u>Per Cent</u>		
	<u>1</u>	<u>2</u>	<u>3</u>
ZrO ₂	--	--	--
SiO ₂	0.19	0.21	0.37
Al ₂ O ₃	0.09	0.35	0.12
Fe ₂ O ₃	0.43		0.75
CO ₂	--	--	--
H ₂ O	0.70	--	not det.
Insol.	0.15	--	--
Rem.	<u>0.31</u>	<u>--</u>	<u>--</u>
Total	100.06	99.85	100.13

1 - Wilberforce, Haliburton County, Ontario. Remnant is He and other gases.

2 - Lac Pied des Monts, Saquenay District, Quebec.

3 - Winnipeg River area, southeastern Manitoba.

NOTE: Thorium dioxide values in uraninite specimens seem to fall within the range 0.00% - 13.94%.

Generally speaking, the thorium dioxide value will be much less than 3.00%.

MINERAL: monazite

Formula: (Ce,La,Y,Th)(PO₄)

Typical Analysis:

<u>Component</u>	<u>Per Cent</u>
CaO	0.35
MgO	0.02

<u>Component</u>	<u>Per Cent</u>
Ce_2O_3	22.63
$(\text{La}, \text{Nd})_2\text{O}_3$	34.63
Y_2O_3	4.66
Fe_2O_3	0.08
Al_2O_3	0.10
ThO_2	7.32
SiO_2	1.54
P_2O_5	27.89
H_2O	0.40
Ign. Loss	--
Rem.	<u>0.65</u>
Total	100.27

Accessory minerals in the ore beds include zircon (ZrSiO_4), leucoxene (Titanium rich alteration product), anatase (TiO_2), rutile (TiO_2), feldspars and titaniferous magnetite ($\text{Fe}_3\text{O}_4 + \text{Ti}$). Sulphides present in variable but generally minute quantities include pyrrhotite (Fe_{1-x}S) ($x = 0 \rightarrow 0.2$), chalcopyrite (CuFeS_2), galena (PbS), sphalerite (ZnS), molybdenite (MoS_2), marcasite (FeS_2 - like pyrite), and cobaltite (CoAsS).

The mineralogical characteristics of the ore conglomerates in the Elliot Lake district are remarkably uniform. The ratio of pebbles to matrix has been estimated to be 65:35, and the composition of the conglomerates of the Quirke Lake

Trough has been estimated to be as follows:

Quartz pebbles	- 40 - 65%
Sulphides	- 2 - 8%
Sericite	- 5 - 20%
Matrix quartz	- 15 - 25%

Variations in the proportions of the radioactive minerals are shown by the wide variation in the ratio of thorium to uranium. Th/U varies from bed to bed, ranging from 0.1:1 to 5:1. However, the uranium content of the orebodies is quite constant and averages between 2.0 and 2.5 lbs. $U_{38}O_8$ per ton. In a typical ore conglomerate, the quartz pebbles are closely sized and range from 0.5 to 1.5 or 2 inches in diameter. They are usually moderately rounded to sub-angular and are loosely to tightly packed. The pebbles consist mainly of silica; white and grey pebbles are predominant, but pebbles of pink quartz, chert and jasper also occur, as well as occasional pebbles of feldspar and granite. The matrix consists of poorly sorted granules and silt-sized particles of quartz, sericite, feldspar and some chlorite. In addition, pyrite, radioactive minerals and other minerals are present here. Except for rare occurrences in fine pyritized fractures in the pebbles, most of the radioactivity is present in the matrix, and the radioactive minerals usually occur in the more richly pyritized areas of the matrix.

Brannerite, uraninite, and monazite are responsible for most of the radioactivity in the conglomerate. Brannerite, a metamict uranium titanate, is

believed to be the most abundant ore mineral in the conglomerates, although either uraninite or monazite may predominate locally. The brannerite occurs as discrete, irregular aggregates of fine prismatic crystals that are often intergrown with quartz as well as with anatase and/or rutile. The average size of the aggregate is approximately 65 to 100 mesh and the maximum size is 20 to 28 mesh. Some of the brannerite also occurs as fine stringers and disseminations in the quartz-sericite matrix.

Uraninite is present as subhedral grains, brecciated grains and remnant crystals that range in size from 48 to 325 mesh, with an average size of about 150 mesh. Some intergrowths of uraninite and brannerite are present.

Monazite, a rare-earth phosphate containing thorium and a minor amount of uranium, occurs as rounded grains with an average size of approximately 65 mesh and typically contains very fine inclusions of pyrite. The monazite is believed to account for most of the thorium and rare-earth contents of the radioactive conglomerate ores.

Several other radioactive minerals have been noted in the conglomerate ores, including zircon, hydrocarbon ("thucholite"), coffinite, pitchblende, and uranothorite, but these minerals contribute only a very small proportion of the total radioactivity of the ores.

Pyrite is the main sulphide mineral in the matrix, pyrrhotite is less abundant and other sulphides are rare. The pyrite is present as euhedral to subhedral grains and in aggregates; the individual masses are usually between 14 and

100 mesh in size. The concentration of pyrite in the matrix varies considerably from place to place.

MINING (1958 - 1964)

Mine Development

On the basis of information obtained from surface diamond drilling and with the knowledge that the shafts had to be on the eastern fringe of the orebody and that the concentrator would be ready as soon as the shafts were down, the Company chose a wholly mobile trackless room-and-pillar mining method for fast development in ore. This method called for 25-foot rooms driven at \pm 10 per cent grade, or just off the strike of the ore, with 25-foot breakthroughs taken at 50-foot centres through the 20-foot pillars. Mobile equipment consisted of three-boom hydraulic jumbos, mounted on tractors, front-end, rubber-tired loaders and 10-ton rear-dump tractor-trailer haulage units.

The ore bed encountered averaged 6 feet in thickness instead of the 11 to 12 feet indicated by surface drilling and thus was not suited to a wholly mobile method.

In February 1959, the method was modified to updip stoping with airleg drills and slushers from 18' x $9\frac{1}{2}$ ' single haulageways. The haulageways were driven in ore with mobile equipment at 280-foot centres from panel entries driven at \pm 12 per cent to the strike of the ore. Overhead muckers, in combination with Torkar shuttlecars, replaced the former combination for mucking these narrow headings. An average advance of 200 feet per month of completed haulageway, plus stope

preparation, was achieved. Since the haulageways developed only 90 tons of ore a foot, more than 1,000 feet of advance was required each month, and thus, six haulageways, at the minimum, were driven on a continuous basis.

Once a safe margin of stope reserves was developed, the method was modified to driving some of the haulageways with airleg drills and slushers by top-slicing the ore and preparing the stopes at first pass and then benching the waste bottom. The broken ore and waste were end-loaded into Le Tourneau trucks or Torkars via a 10-foot-long stope chute set on posts.

The pattern of slusher stoping underwent several modifications and several methods were tried. The final stoping layout for updip slashing to a pilot or slot raise proved to be the most satisfactory for the conditions prevailing at the Stanrock property. The standard stopes were 45 feet wide and 220 feet long with 15-foot pillars top and bottom and between stopes (roof-support problems frequently necessitated a stope width narrower than 45 feet). The normal stoping sequence consisted of driving a pilot raise up along the pillar wall and then slashing the remainder of the stope to full width. The slash was carried in two or three steps to avoid holdback and to give better slushing characteristics. All rounds and slashes were primed with short-period electric caps and the blasts initiated with 50-shot plunger machines.

Where the ore thickness exceeded 8 feet, a top slice was mined up the complete stope and the remaining ore removed by either downhole or uphole benching.

A normal stope crew comprised four men -- two drillers on one shift, drill-

ling an average of 176 feet each, and a slusherman and a rock-bolter on the second shift who slushed 100 tons of ore and installed 16 to 20 rock bolts. Rock-bolting was done on a 3' x 4' pattern with 5/8" x 5' high-tensile rock bolts with left-hand thread at both ends and a bail-type shell. Eight-foot rock bolts were used for renovating blocky ground or in the vicinity of faults or dykes. More than 10,000 rock bolts were installed per month by means of a hand-held "T" wrench for torquing to 160 foot-pounds. Stope productivity in 1961 was maintained at 25 tons per man per shift.

Four-man crews were also employed in jumbo drives -- two drillers on one shift and a mucking machine operator and a rock bolter on the second shift. The average advance per jumbo round was 10.8 feet.

All broken ore was hauled by trucks or Torkars to one of six grizzlies above conveyor feed points. The grizzly openings were 20" x 20" with one or two 30-inch slots which passed maximum pieces of about 900 lbs. These pieces generally broke up in the drop to the chute or at transfer points. The ore was fed onto the conveyor belts by means of air-operated ball-chain gates and vibrating feeders.

The conveyor system, which was located mainly in the footwall of the ore, comprised three 48-inch belt conveyors, each approximately 1,500 feet long. The normal carrying capacity of the conveyor system was 300 tons per hour with a surge capacity of approximately 600 tons per hour. The muck storage capacity of the system was 3,000 tons, of which 1,800 tons were at the loading pocket at No. 1 shaft.

Services and Equipment

(i) Ventilation

The main supply fans were underground at No. 2 shaft and consisted of five, three-stage, direct-driven, contrarotating fans in parallel. The total rated capacity of the system was 200,000 cfm at 8 inches of static water pressure. During the winter, the natural ventilation added 60,000 cfm. This push system required a good substantial **air-lock** entry into the mine and air-operated doors with adequate signals, and these were incorporated at Stanrock.

About 25 booster and auxiliary electric fans were in use and ranged from 35,000 cfm to 10,000 cfm in capacity.

The total installed air-heating capacity of 13,300,000 Btu per hour was designed to heat 210,000 cfm of air from -20°F to $+35^{\circ}\text{F}$.

(ii) Compressed Air

Three 3,300 cfm compressors supplied compressed air to the mine at 110 psi through a 10-inch main down No. 1 shaft and a 6-inch main down No. 2 shaft. These mains inter-connect underground. An 8-inch loop carried compressed air to 6-inch and 4-inch lines installed in headings and to 2-inch lines leading into individual stopes.

(iii) Drill Water

Water for drilling and for wetting down the muck was supplied through a 6-inch line underground via a 3-inch float-control system in No. 2 shaft. The 6-inch line extended about 4,000 feet in the upper-sheet orebody and, from there,

the water was fed down through 4-inch and 2-inch arteries to the lower production areas.

(iv) Pumping and Drainage

In 1961, the mine was pumping approximately 200,000 imperial gallons of water per day - the water coming generally from the drilling and washing operations. This represented about 40 per cent of the pumping capacity.

The main settling and clear-water sumps were at No. 1 shaft. Seventy per cent of the production-area water was collected below the main sump elevation and pumped from a secondary sump system with a 2 MRV-40 centrifugal pump. A similar pump was installed as a stand-by. The water was pumped from the main clear-water sump by a six-stage pump with a rating of 410 gpm at a 1,040-foot head. A second pump was installed as a stand-by. Because of the low pH of the mine water (3.0 or less), a caustic solution was fed down No. 1 shaft via a 3-inch pump line that was formerly used by the contractor and was metered into the intake of the pump through a solenoid valve triggered by the starting and the stopping of the pump. The drainage water was pumped to the 2,100-foot level where it was relayed to the 1,100-foot level and thence to the surface by similar pumps. Only one stand-by was kept available for the pumps at the 1,100 and 2,100-foot levels. The drainage water went into a 2,500 gallon receiving tank on the surface from where it was pumped to the mill for the recovery of the contained uranium values.

(v) Shaft Construction

Both shafts (No. 1 and No. 2) are 20'8" x 9'7" outside the timbers.

No. 1 is a three-compartment shaft with two 6'0" x 8'0" skip compartments (15 ton skips) separated by a 5'6" x 8'0" manway and pipeway. No. 2 shaft has a 12'0" x 8'0" cage compartment and a 5'6" x 8'0" manway and pipeway. No. 2 shaft (used for hoisting men and materials) has a single-deck cage of 48 man or $9\frac{1}{2}$ ton capacity.

(vi) Drilling and Blasting

Production drilling was done with 2 $5\frac{1}{8}$ -inch-bore airleg drills equipped with 1.42-inch bits. The drills were made of $7\frac{7}{8}$ -inch hexagonal steel and were 8 feet long for pilots and 10 feet long for slashes. The average drilled per machine shift was 176 feet. Drilling per consumed unit was 150 feet per drill bit and more than 400 feet per drill steel. There were 150 airleg drills in service and 56 stoper drills for rock-bolting. All blast holes were pneumatically loaded with an ammonium nitrate-fuel oil mix and collar-primed with short-period caps with 4-foot leads and one or two sticks of 1-inch or $7\frac{7}{8}$ -inch diameter Cilgel "C" high explosive.

Until March in 1962, four Gardner-Denver, three-boom, hydraulically-operated drill jumbos (three of them mounted on tractors) were employed on stope-development drives. However, near the end of conventional mining, only one jumbo was in use.

Drills of a $3\frac{1}{2}$ -inch bore were used with 1 $5\frac{1}{8}$ -inch or 1 $3\frac{3}{4}$ -inch bits on 13'0" x 1 $1\frac{1}{8}$ " round lugged steel. On the average, the bits lasted for 90 to 100 feet. The drill steel provided more than 400 feet of drilling per piece. Two men alternating from day-to-day on the controls and front-end work took the

18' x 9 $\frac{1}{2}$ ' rounds with 55 holes. All rounds were primed with short-period caps and all but the wet down-grade headings were blasted with an ammonium nitrate-fuel oil mixture.

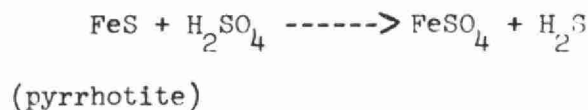
(vii) Bacterial Leaching (1960 - 1970)

As high-grade ore reserves were depleted, mining emphasis gradually shifted (beginning in 1960) from conventional underground practice to uranium extraction by bacterial leaching.

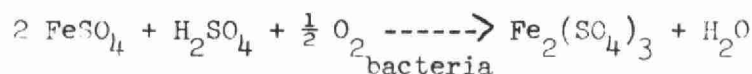
Conventional underground mining ceased at the end of October in 1964.

The chemical reactions involved and the function of bacteria in the leaching of Elliot Lake uranium ores may be represented by the following equations:

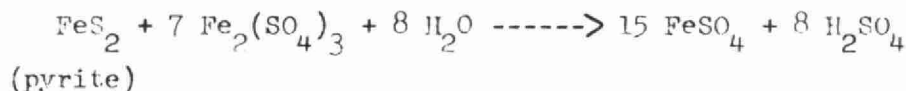
(Eqn. 1)



(Eqn. 2)



(Eqn. 3)



(Eqn. 4)



(Eqn. 5)



In this series of equations, the bacteria (Ferrobacillus - Thiobacillus group) are assumed to take part only in promoting the oxidation of ferrous iron to the ferric state as shown in Eqn. 2. The subsequent production of sulphuric acid and the incidence of uranium dissolution are considered to be the results of reactions between ferric iron and pyrite, and ferric iron and uranium minerals (Eqn. 5), involving the reduction of ferric iron to the ferrous state.

The past practice at Stanrock has been to return all barren from the mill circuit to the mine. As a result, the ferric iron concentration in the mine water has built up to a point where, presently, great emphasis is apparently no longer placed on the action of bacteria in the uranium dissolution process in the mine itself. The ferric iron concentration in the barren may, in the near future, build to an undesirable level with the result that some barren may have to be bled off to the tailings area.

NOTE: The mine water coming into the mill is presently being treated with H_2SO_4 . The most noticeable effect of the treatment is the apparent reduction of the concentration of the in-coming suspended ferric iron. Apparently, if the high suspended solids concentration was not reduced in this manner, the solids would interfere in the subsequent ion exchange process.

Mine Water to Mill (from underground)

	Concentration ppm
Total Solids	6170
Susp. Solids	15
Diss. Solids	6155
Sulphates as SO_4	2800
Nitrogen (Nitrate)	102
Nitrogen (Ammonia)	100

	Concentration ppm
Chloride as Cl	621
COD	50
pH at Lab.	2.3
Acidity as CaCO_3	3180
Iron as Fe	280
Titanium as Ti	0.00

Barren to Mine (i.e., return underground)

	Concentration ppm
Total Solids	6540
Susp. Solids	10
Diss. Solids	6530
Sulphates as SO_4	3300
Nitrogen (Nitrate)	105
Nitrogen (Ammonia)	150
Chloride as Cl	773
COD	39
pH at Lab.	2.0
Acidity as CaCO_3	3620
Iron As Fe	272
Titanium as Ti	0.4

MILLING

(i) Past Milling Procedures

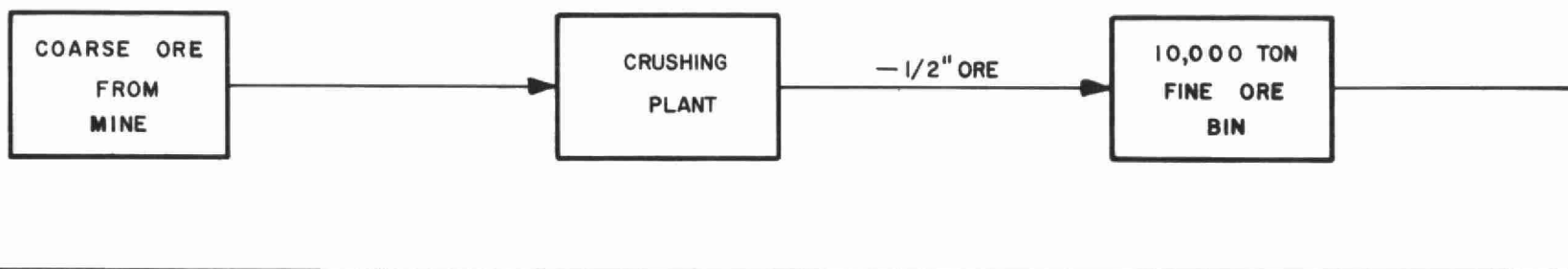
The following is the general mill flowsheet (1958) for Stanrock Uranium Mines Limited. The inclusion of the original flowsheet in this report is intended as a guide to the methods of milling and subsequent uranium recovery employed at Stanrock in its initial years. The flowsheet also serves the purpose of defining, in general, the points of addition of the various reagents used in the milling of the ore. The flowsheet, however, in no way takes into account the various modifications of the circuit that may have taken place during later stages of mine-mill development.

Reagents generally added during the milling of Elliot Lake ores are as follows:

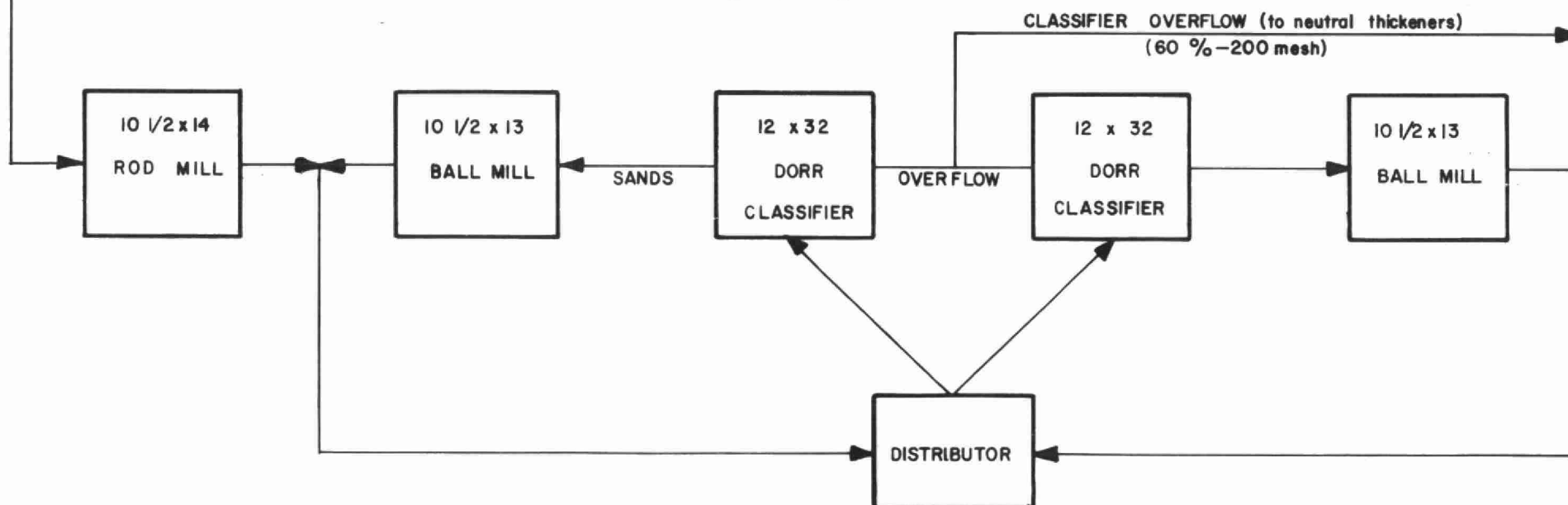
<u>Reagent Added</u>	<u>Amount - lbs./ton Ore Milled</u>
Sulphuric Acid	67-100
Sodium Chlorate	2-4
Lime as CaO	33-46
Nitric Acid	3-5
Sodium Hydroxide	1-2
or	
Magnesium Hydroxide	0.5
Glue	0.3-1.6
Separan	0.03-0.05
Water	1.8-3.3 tons per ton ore milled.

STANROCK URANIUM MINES LTD.

CRUSHING

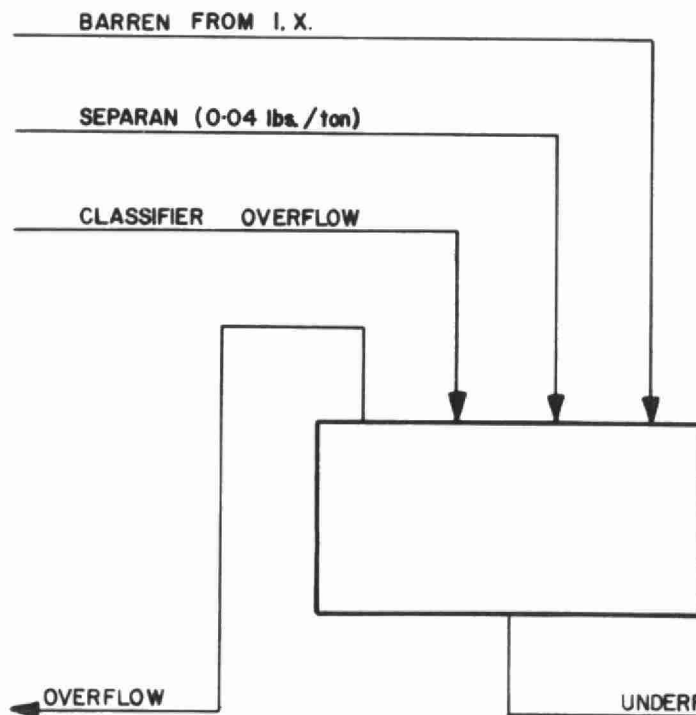


GRINDING



STANROCK URANIUM MINES LTD

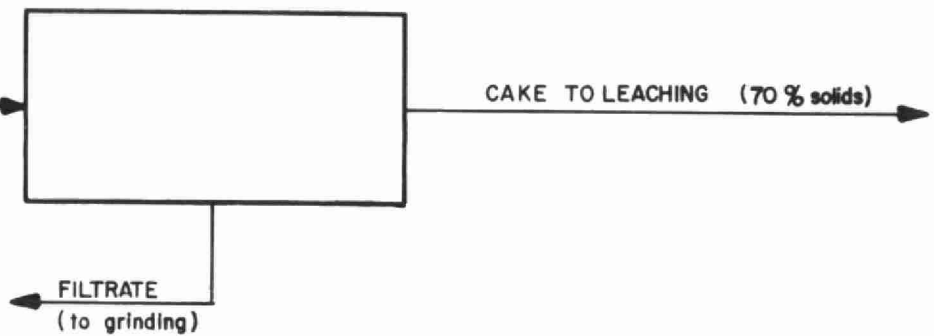
NEUTRAL THICKENERS



UNDERFLOW

OVERFLOW
(to grinding plant)
PH = 6.8

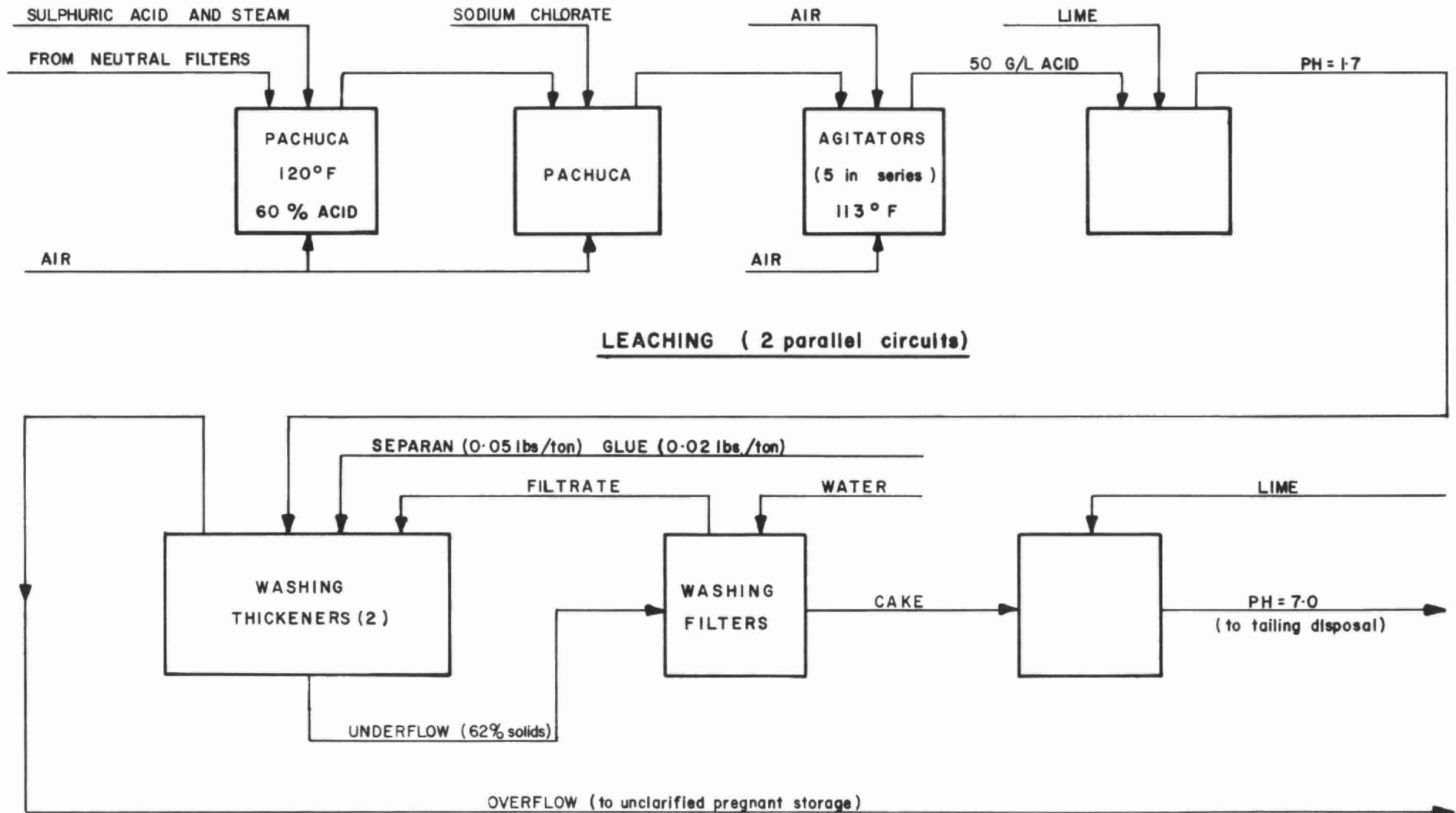
NEUTRAL FILTERS



FILTRATE
(to grinding)

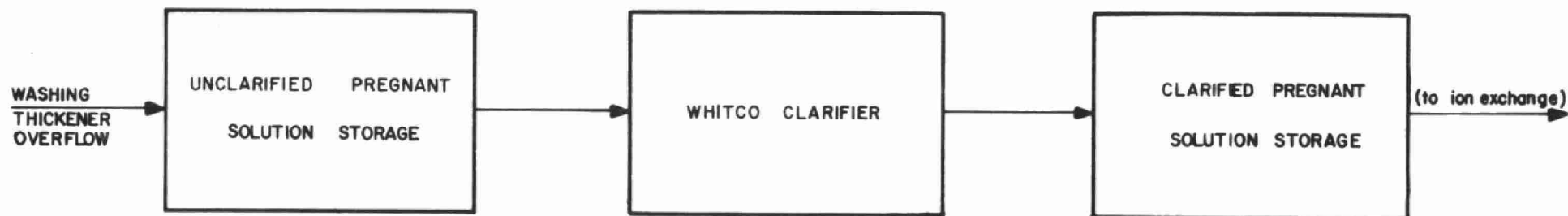
CAKE TO LEACHING (70 % solids)

STANROCK URANIUM MINES LTD.

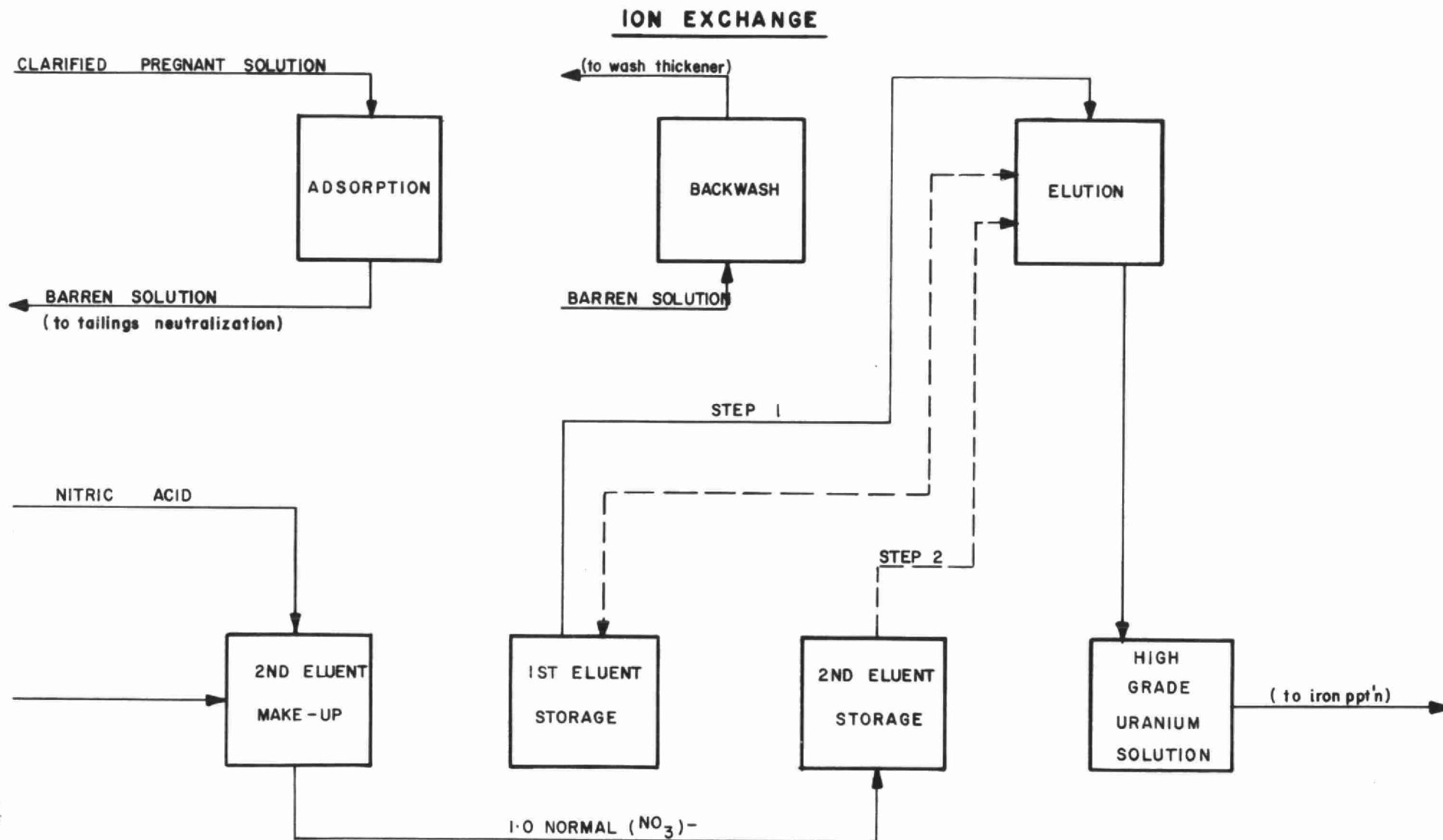


STANROCK URANIUM MINES LTD.

DOUBLE CLARIFICATION

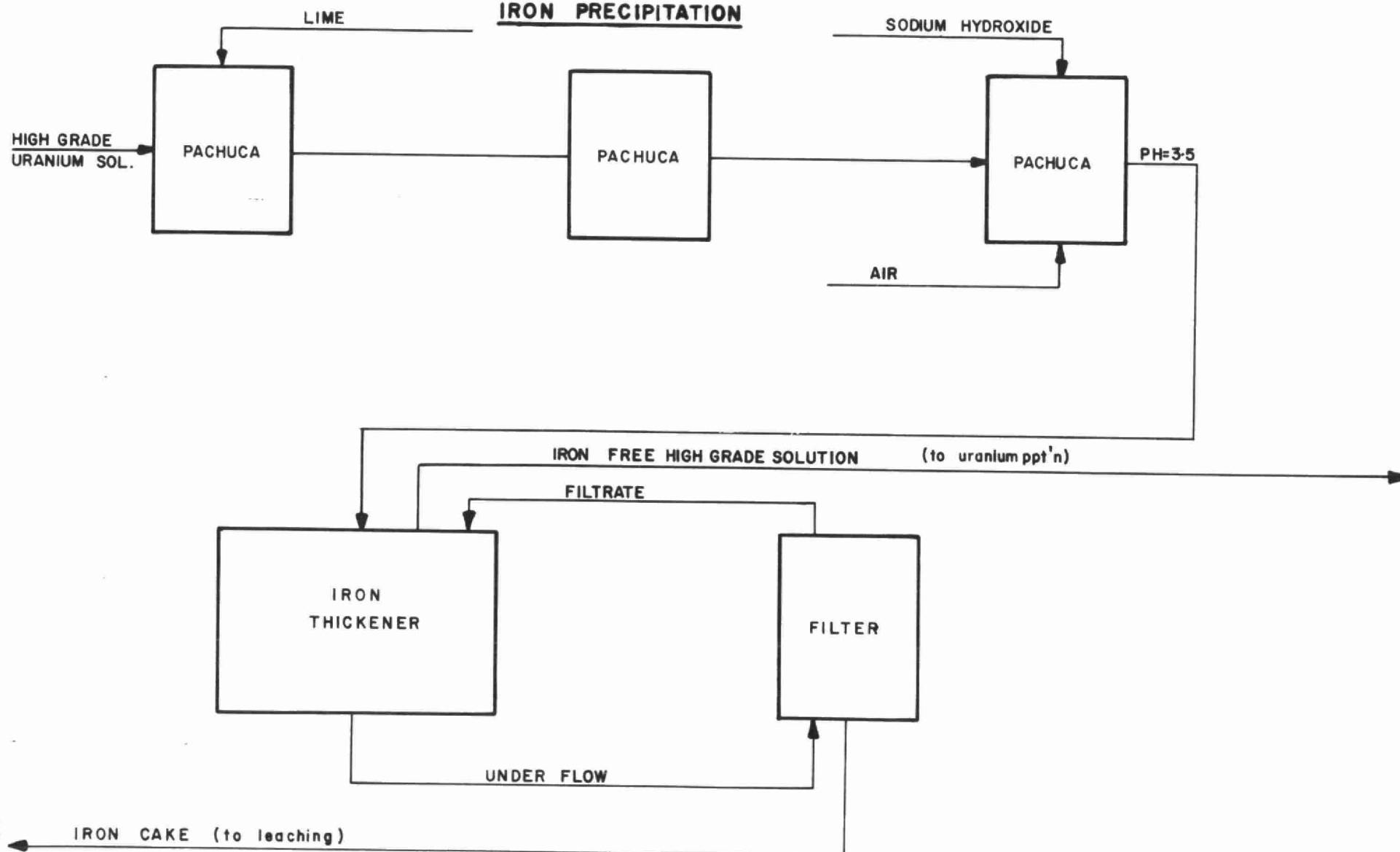


STANROCK URANIUM MINES LTD.



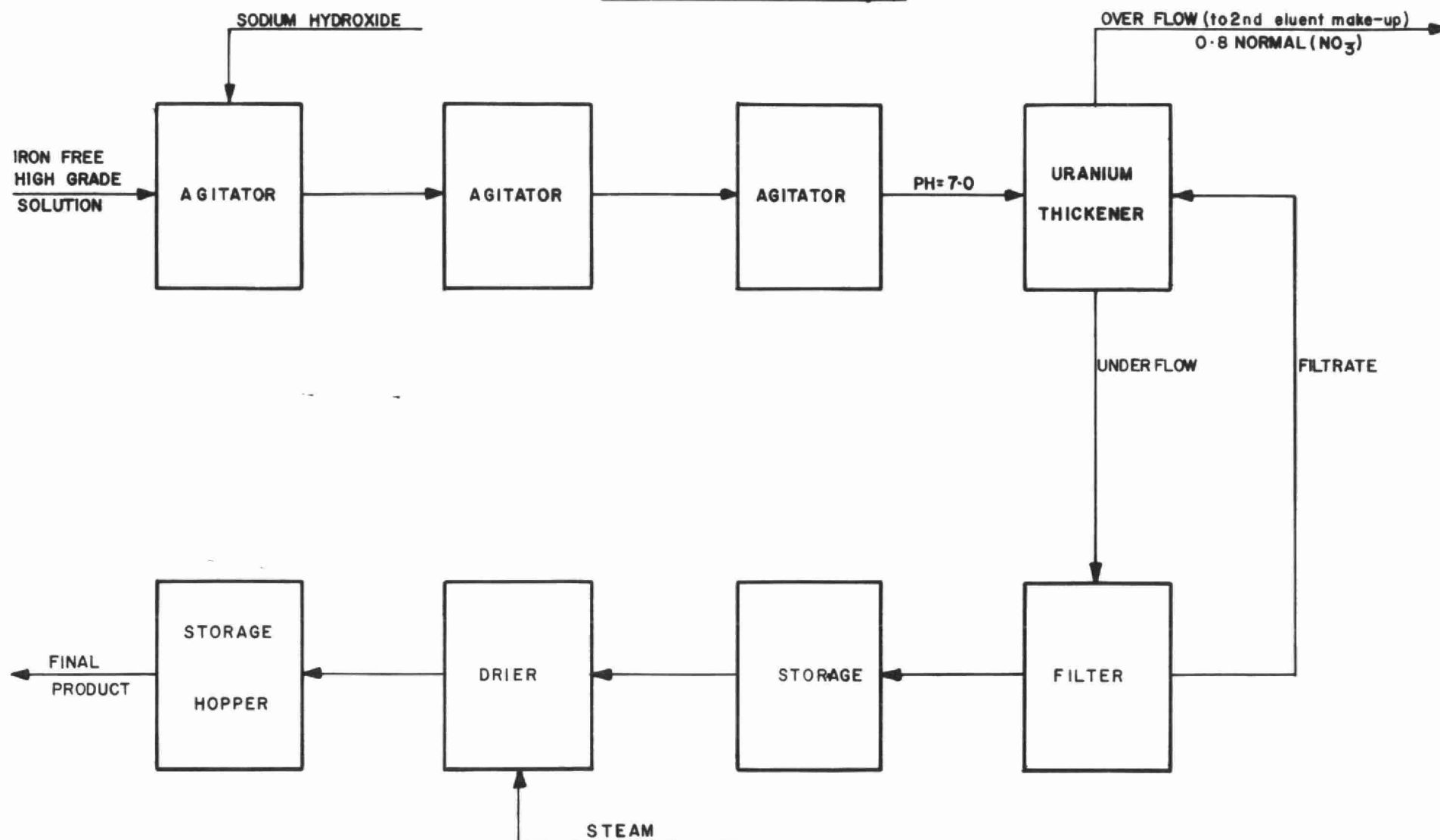
STANROCK URANIUM MINES LTD.

IRON PRECIPITATION



STANROCK URANIUM MINES LTD.

URANIUM PRECIPITATION



(ii) Present Milling Procedures (1969)

The following mill description (see Fig. 1) refers to the circuit as it presently exists (December 1969). The circuit is quite simple due to the fact that no broken ore is being recovered from the mine. The mill feed consists solely of recirculated underground mine water.

The mine water, with a pH of approximately 2.4 and containing from 0.1 to 0.15 gms/litre of U_3O_8 , first passes through a clarifier with the overflow being directed to a battery of ion exchange columns. The barren (0.0001 gms U_3O_8 /litre, + ThO_2 , + R_2O_3 --- pH 2.0) from the ion exchange columns is presently being returned to the mine. In the near future, some or all of the barren may be treated to recover the contained thorium and rare-earth values (depending entirely on the fluctuating industrial demand for these materials) or the barren may be simply discharged to the tailings area as waste. The uranium rich eluate (5 gms U_3O_8 /litre) from the ion exchange columns is pumped to a tank where iron is precipitated by ammonia addition. The clear overflow from the tank is again treated with ammonia to cause the precipitation of the uranium values: $6 NH_3 + 6 H_2O + 2 UO_2SO_4 \rightarrow (NH_4)_2U_2O_7 + 2 (NH_4)_2SO_4 + 3 H_2O$. The precipitate is filtered and steam dried. The resultant material (yellow cake - 80 per cent U_3O_8) is packed in drums capable of holding 500 lbs. of product. The drums are shipped out as required.

NOTE: The iron sludge underflow (if any) from the various tanks mentioned is accumulated until the total quantity justifies treatment for the entrained uranium values. Alternatively, the iron sludge may be treated and recycled underground (via ion exchange) or simply discharged to the tailings area.

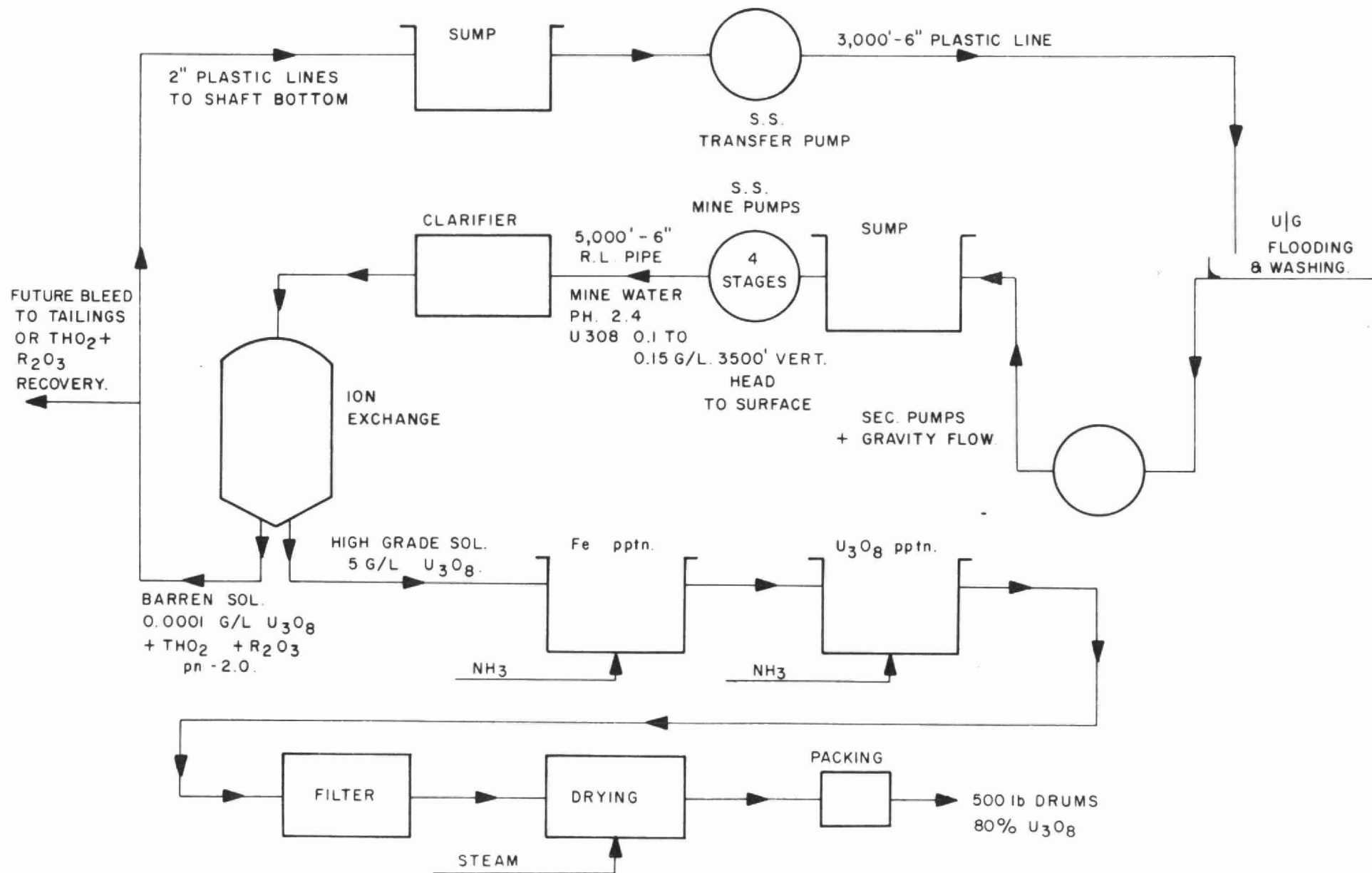


FIG. 1- STANROCK URANIUM MINES
MINE WATER TREATMENT

PRODUCTION DATA

<u>Year</u>	<u>Ore Milled (tons)</u>	<u>Mill-Head Grade (lbs. U_3O_8/ton)</u>	<u>U_3O_8 Produced or Shipped (lbs.)</u>
1958	n.a.	n.a.	950,000
1959	1,000,000 e	n.a.	2,126,000
1960	1,000,000 e	n.a.	2,150,000 e
1961	1,036,937	2.01	2,132,605
1962	1,060,163 *	1.902 *	1,900,145 *
1963	1,033,515	1.889	1,845,662
1964	n.a.	n.a.	1,549,558
	end of conventional mining		
1965			
1966			
1967	bacterial leaching only		
1968			
1969			

n.a. - not available

e - estimated

* - Northern Miner, January 24, 1963

To September 30, 1966, a total of 11,798,778 pounds (Canadian Mines Handbook - 1968-1969) of U_3O_8 had been recovered. The total value of this production (plus yttrium oxide) was \$122,418,424.

For the twelve months ending December 31, 1967, Stanrock had gross sales of \$434,898 and sustained operating costs of \$644,960 leaving an operational loss of \$210,062. After adding other income \$36,373, machinery and equipment disposals \$16,132, net profit from operation and sales of the Fulfillment Division \$138,431 and deducting other expenses, the net loss was reduced to \$26,026.

This is a contrast to previous years:

<u>Year</u>	<u>Sales</u>	<u>Operating Profit</u>	<u>Net Profit</u>	<u>Net Loss</u>
1965	--	\$48,957	\$796,055	--
1966	\$818,681	46,978	134,641	--
1967	434,898	--	--	\$26,026
1968		not available		
1969 *	339,189	--	--	38,149 *

* first 9 months 1969 only

During 1969, approximately 30 people were employed at Stanrock's Elliot Lake property.

COMPANY PROPERTIES

- (1) 19 claims, 3 fractions, 615 acres, Townships 144 and 150, Quirke Lake sector, Elliot Lake, Ontario.
- (2) Stanward Corp. assets (acquired in mid-1967), 138 claims adjoining the Stanrock claims in Elliot Lake, Ontario.
- (3) 69.5 per cent interest in Stanatomic Uranium Mines which holds a 31 claim uranium prospect northeast of Lake Nordic near Elliot Lake,

Ontario (inactive).

- (4) Late in 1967, Stanrock obtained option from North Rock Explorations and Green Valley Mines to acquire a 51 per cent interest in a group of 17 claims situated about two miles from the Stanrock mine.

Ore Reserves

In mid-1968, Stanrock Uranium Mines Limited completed a re-assessment of reserves at the Stanrock mine and estimated the property to contain approximately 10,000,000 tons of assured and probable uranium-bearing material grading about 1.5 lbs. U_3O_8 per ton, plus large additional possible reserves. These deposits are not being depleted by the bacterial leaching program.

Drilling on the optioned 17 claim group (see Section 4 under "Company Properties" above) indicated 750,000 tons of mineralized material grading 1.35 lbs. U_3O_8 per ton across 10 feet in the northwest corner of the property. A further 4,250,000 tons of material grading just under 1 lb. U_3O_8 per ton across 11 feet was located to the east.

SALES CONTRACTS

- (1) Contract with Eldorado Mining and Refining Ltd. for the delivery of 9,198,000 lbs. of U_3O_8 worth \$95,000,000 - to March 31, 1963.
- (2) September 1962 to May 1966 - delivered 2,886,105 lbs. of U_3O_8 (\$5.50/lb.) on behalf of a United Kingdom government contract.
- (3) In 1966, made deliveries of U_3O_8 to Eldorado (at \$4.90/lb.).
- (4) In 1967, all U_3O_8 sales made to reactor mfr. at price of better than \$5 per lb.

- (5) From January 1966, three-year contract made with Michigan Chemical Corp. of Chicago, Illinois, for sale of yttrium oxides.

NOTE: Beginning in early 1966, Stanrock began producing a rare-earth concentrate. However, by the middle of 1967, rare-earth production was suspended due to unfavourable market conditions. The production of Y_2O_3 concentrate has not been resumed to date (December 1969).

NOTE: Contract #4 (above) was a three-year contract arranged by the Company which called for the delivery of 7,500 lbs. of U_3O_8 per month.

All contracts are due to expire in June of 1970 and, to our knowledge, the Company has not been successful in securing further contracts.

WATER SUPPLY

(a) Domestic Supply

All domestic water used at the Stanrock mine, mill, and townsite is pumped from a small spring-fed lake (Nelson Lake) located near the Stanrock mill. Treatment is affected by means of a gas-type chlorinator located in the pumphouse adjacent to the Lake. A standby hypochlorinator is also tied into the system. Water is carried to the mine-mill complex via a 4-inch diameter cast iron main. Early in 1969, there were approximately 75 people being served by the water from Nelson Lake; 50 in the townsite living in 18 homes, and about 25 mine-mill employees.

The following data is pertinent to the domestic water supply from Nelson Lake:

- (i) setting on chlorinator - 1 lb. per 24 hours
- (ii) residual at pumphouse - 0.5 ppm
- (iii) average residual at mill- 0.25 ppm

(iv) pumping capacity

(a) main pump (25 hp electric drive)

- 400 gpm

(b) standby pump (40 hp electric drive)

- 400 gpm

NOTE: There are no water meters on the system.

Nelson Lake - Raw Water

<u>Year</u>	<u>pH</u>	<u>Iron as Fe</u>	<u>Chlorides as Cl⁻</u>	<u>Sulphate as SO₄</u>	<u>Kjeldahl Nitrogen</u>	<u>Hardness as CaCO₃</u>	<u>Alkalinity as CaCO₃</u>	<u>Turbidity Units</u>
1967	7.1	0.35	12	--	0.07	48	29	0.6
1967	7.0	0.10	7	--	0.71	44	34	2.6
--	--	--	--	--	--	--	--	--
1969	7.4	0.35	7	13	0.46	46	29	2.8

NOTE: All results except pH are reported in parts per million unless otherwise indicated.

(b) Industrial Supply

The mine-mill operates essentially as a closed circuit. Approximately 2,500 tons of water are pumped each day but this figure represents only recirculated mine water. A daily influx of 25 tons of fresh water (pumped from Quirke Lake) is used for process make-up.

Quirke Lake Intake Water at Stanrock Mill

<u>BOD</u>	<u>COD</u>	<u>pH</u>	<u>Chlorides</u> <u>as Cl⁻</u>	<u>Sulphate</u> <u>as SO₄</u>	<u>Total</u>	<u>Solids</u> <u>Susp.</u>	<u>Diss.</u>	<u>Iron</u> <u>as Fe</u>	<u>Acidity</u> <u>as CaCO₃</u>
1.8	8	4.1	17	105	190	5	185	?	14

NOTE: All results (except pH) expressed as ppm.

TAILINGS DISPOSAL - CHRONOLOGICAL HISTORY

The present Stanrock tailings area occupies the natural basin of a small lake that originally discharged to Halfmoon Lake via "Cinnabar" Creek and "Orient" Creek (see Sheet #1). Aerial photographs of this lake are to be found in the files of the Ontario Water Resources Commission. The lake appears to have been much larger than Beaver Lake.

When mining activities commenced in the area, the natural depression was initially used as a tailings area by two mining companies (Can-Met and Stanrock). Later, however, Stanrock Uranium Mines Limited became the sole contributor of mining wastes to the basin.

1957

On October 21, 1957, Can-Met Explorations Limited (see Appendix A) began production at an initial rate of 900 tons/day. All tails were discharged to the "Stanrock" tailings area. The supernatant liquid was decanted through two concrete decant wells which were connected to a common concrete rectangular pipe that passed (and still passes) through the "South" dam. Waste flow was directed to the Lower Serpent River system.

1958

On March 1, 1958, Stanrock Uranium Mines Limited went into production at an initial rate of 1,000 tons/day. By the year's end, production had increased to 3,000 tons/day (capacity rate). All tails were discharged to the Stanrock tailings area.

1959

Both mills continued to discharge tailings to the combined tailings area. Operations were more or less normal.

1960

On March 24, 1960, Can-Met Explorations amalgamated with Consolidated Denison Mines Limited to form a new Company: Denison Mines Limited. The remainder of the Can-Met contract was transferred to Denison. Milling operations at Can-Met ceased on April 15, 1960.

Thus, from April 15, 1960 onwards, Stanrock Uranium Mines Limited was the only operation to discharge tails to the basin.

1961 - 1963

During these years, conventional mining and milling procedures were followed on a more or less continuous basis. All tailings were discharged to the tailings area.

1964

On April 17, 1964, a section of one of the tailings dams collapsed allowing nearly 9,000 tons of waste material to gain access to Quirke Lake.

On June 9, 1964, a decant well failed. 500,000 tons of tailings escaped to the system of the Lower Serpent River. Conventional underground mining at Stanrock ceased at the end of October in 1964.

1965 - 1969

Uranium recovery at the Stanrock mine continued but was due solely to recovery by bacterial leaching. Tails were not being discharged to the tailings area.

Tailings Area - Mining Claim Boundaries

(see Sheet #2 - back pocket)

Ore deposits of large aerial extent quite commonly force an unusual number of somewhat unrelated mining companies to co-exist on a limited surface area. It would therefore not be too unusual to find that the tailings disposal area of any one particular company actually straddled the claims of another.

In the case of the Stanrock tailings area, the situation is overly complex. Only 3 per cent of the surface area of the tailings basin lies on "Stanrock" claims. The remainder of the surface area is divided unequally among four other claim holders - specifically "Denison" (Denison Mines Limited), "Norgold" (Norgold Mines Limited), "Mentor" (Mentor Exploration and Development Co. Ltd.) and "MacGregor" (?).

SOURCES OF LIQUID WASTES

(a) Mine-Mill Complex

As stated before, the mine-mill complex operates as a closed circuit.

Wastes (liquid or solid) are presently not being discharged to the tailings area. Barren solution from ion exchange is returned to the mine (underground).

Water, withdrawn from Quirke Lake, is used to cool the large air compressors that supply pressurized air to the mine-mill complex. This water (originally of very poor quality because of the effect of the tailings area discharges on the adjacent areas of Quirke Lake) is simply allowed to flow back down the slope from the mill to Quirke Lake.

Compressor Cooling Water (October 22, 1969)

<u>Total</u>	<u>Solids</u> <u>Susp.</u>	<u>Diss.</u>	<u>pH</u>	<u>Zinc</u> <u>as Zn</u>	<u>Nickel</u> <u>as Ni</u>	<u>Sulphate</u> <u>as SO₄⁼</u>	<u>Cobalt</u> <u>as Co</u>	<u>Chlorides</u> <u>as Cl⁻</u>	<u>Hardness</u> <u>as CaCO₃</u>
150	5	145	3.9	0.09	0.17	340	0.0	8	102

NOTE: All results (except pH) expressed as ppm.

Since this water is not chemically changed during its use as a cooling medium, and since the quality of this water simply reflects the quality of the water in the Quirke Lake basin, no detailed comments are required. An improvement in the chemical condition of Quirke Lake will automatically force an improvement of the quality of this intake water.

Technically speaking, however, the compressor cooling water, in this case, should be neutralized with a chemical base (lime, caustic, etc.) before it is discharged to Quirke Lake. The neutralization should be continued until the intake water exhibits the pH characteristics of the many uncontaminated bodies of water in the area.

(b) Townsite

Domestic wastes from the townsite (housing approximately 50 people) are discharged to the Stanrock tailings area. The discharge line has recently been moved nearer to the decant structure in an attempt to force the wastes to leave the tailings area via the decant structure rather than as seepage through the dams.

(c) Tailings Area

Important: See Sheet #1 in the back pocket of this report.

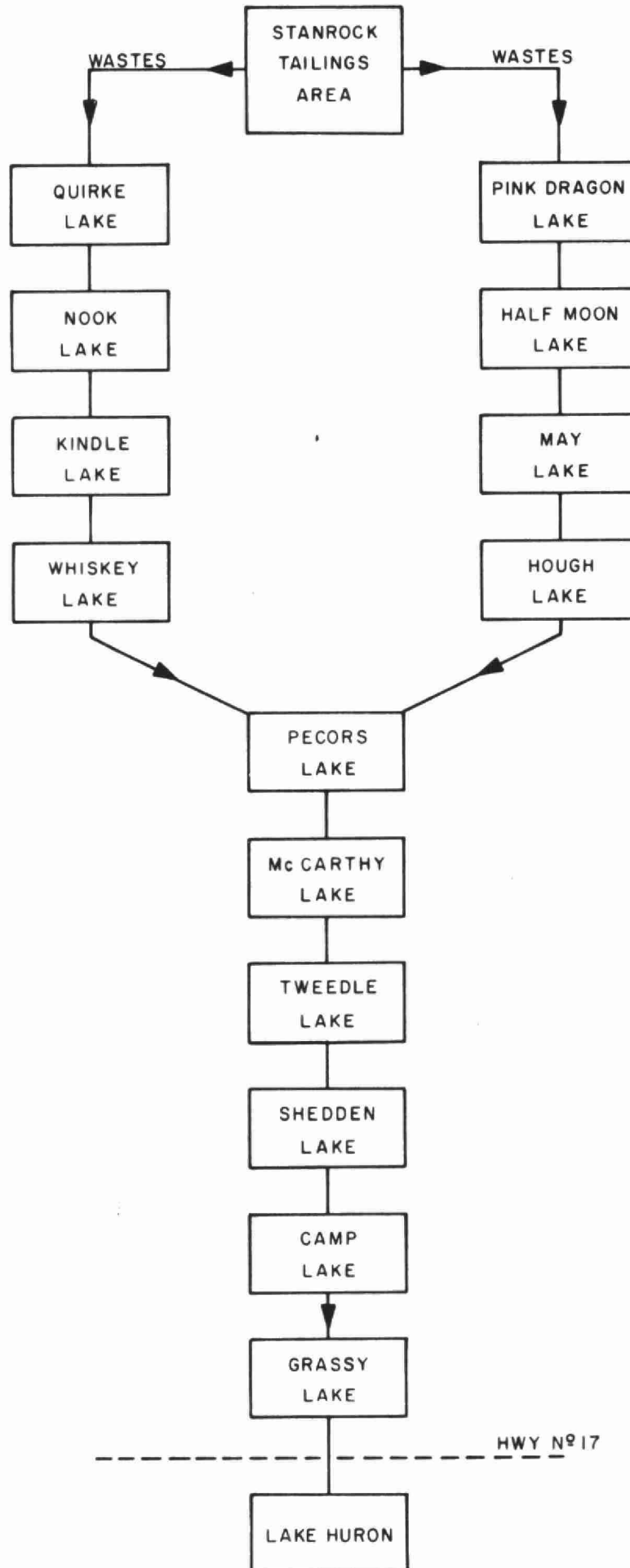
The Stanrock tailings area is the largest and most complex single source of pollution in the Elliot Lake district. Because of the geographical position of the tailings body, highly contaminated wastes gain access not only to Quirke Lake but also to the Lower Serpent River system.

In the summer of 1969, the Ontario Water Resources Commission staff was finally able to define the intricate drainage system of the Stanrock tailings mass. The tailings area decant along with eight major seepage flows were found to account for the entire effluent from the tailings area.

The following is a precise description of each flow:

NOTE: The names "Orient Creek", "Cinnabar Creek", "Swallow Hill" and "Pink Dragon Lake" are unofficial. They were adopted for use when no official designations could be found.

NOTE: The radiological characteristics of each flow are discussed in the section entitled "Radiological Characteristics".



(A) Decant Overflow

Photograph A

Decant overflow as seen from the top of the South dam. The wastes are flowing toward the horizon.

(1) Flow:

Variable but averages 150,000 gpd.

(2) Physical Description of Waste:

Pale yellow solution generally slightly turbid.

(3) Chemical Characteristics of Tailings Area Decant:

	Concentration ppm	
	<u>June 18, 1969</u>	<u>October 21 & 22</u>
Total Solids	13,460	
Susp. Solids	25	
Diss. Solids	13,435	
Sulphates as $\text{SO}_4^{=}$	6,900	2,440
Total Iron (as Fe)	368	220-340
Total Diss. Iron		
Total Ferrous Iron		90-156
Total Ferric Iron		
Nitrogen (Nitrate)	10.0	
Nitrogen (NH_3)	100	
Chloride as Cl^-	97	
COD	278	
pH (at Lab.)	2.0	2.8
Uranium as U	5.6	3.2
Zinc as Zn		9.4
Nickel as Ni		1.5
Cobalt as Co		1.5
Copper as Cu		2.2
Manganese as Mn		3.6
Titanium as Ti	3.6	
Acidity as CaCO_3	7,700	

(4) Direction of Flow:

The decant overflows to Cinnabar Creek. Cinnabar Creek meets Orient Creek after crossing the mine road. Orient Creek flows through the "spill area" and discharges to Pink Dragon Lake. Pink Dragon Lake overflows to a small stream

(a continuation of Orient Creek) that in turn flows into Half Moon Lake.

(B) Decant Swamp Overflow

Photograph B

The Decant Swamp - a peculiar circular basin perched 40 feet above Cinnabar Creek. This basin located within 100 feet of the tailings area receives seepage from the nearby tailings area and also intercepts wind-blown ferruginous particles.

(1) Flow:

Generally negligible (3-5 gallons per minute) but during heavy precipitation may reach 5,000 gpd.

(2) Physical Description of Liquid Waste:

The decant swamp liquid is a clear dark amber-yellow solution.

Because of the colour, it is generally impossible to see the bottom of the basin where the water is over 2 feet deep.

(3) Chemical Characteristics of the Decant Swamp Liquid:

October 21, 1969

Iron as Fe		Sulphate as	pH	Uranium	Zinc			
<u>Total</u>	<u>Ferrous</u>	<u>SO₄</u>	<u>at Lab.</u>	<u>as U</u>	<u>as Zn</u>	<u>Co</u>	<u>Ni</u>	<u>Cu</u>
420	16.5	2800	2.4	1.9	1.37	1.3	1.0	0.67

NOTE: All results (except pH) expressed as ppm.

(4) Direction of Flow:

The Decant Swamp overflows to the headwaters of Orient Creek.

Orient Creek ultimately discharges to Half Moon Lake.

(C) South Dam Seepage

Photograph C

The South Dam Seepage as it emerges from beneath the South Dam. The size of the seepage flow may be judged from the size of the automobile in the background.

(1) Flow:

Approximately 50,000 gpd. The volume of seepage from this particular source has been steadily increasing. Part of the flow is probably derived by downward percolation from the nearby tailings decant pond.

(2) Physical Description of the Waste:

The seepage here is in the form of a clear light-yellow solution.

(3) Chemical Characteristics of the South Dam Seepage:

Iron as Fe		Sulphate as SO_4	pH at Lab.	Uranium as U	Zn	Co	Ni	Cu
<u>Total</u>	<u>Ferrous</u>							
1530	50.0	6280	2.2	7.4	10	3.1	2.3	2.1

NOTE: All results (except pH) expressed as ppm.

(4) Direction of Flow:

The South Dam Seepage flows into Cinnabar Creek within a few hundred feet of the base of the South Dam. Cinnabar Creek flows into Orient Creek.

(D) Mine Road Seepage

(1) Flow:

Variable but generally within the range 25,000 - 50,000 gpd.

(2) Physical Description of the Waste:

The mine road seepage emerges from the tailings area as a pale-yellow clear solution. Where the stream is over 1 foot deep, the waste appears to be deep amber-red in colour.

(3) Chemical Characteristics of the Mine Road Seepage:

Iron as Fe		Sulphate as SO_4	pH at Lab.	Uranium as U	Zn	Co	Ni	Cu
<u>Total</u>	<u>Ferrous</u>							
120	3.50	760	2.7	0.4	0.98	0.40	0.45	0.26

NOTE: All results (except pH) expressed as ppm.

(4) Direction of Flow:

The Mine Road Seepage arises at the base of the tailings mass near

the mine road itself. The stream cuts across the road, forms a swampy area, and then cuts back across the road to meet Cinnabar Creek. Cinnabar Creek flows into Orient Creek.

(E) Orient Creek Swamp

(1) Flow:

Approximately 100,000 gpd.

(2) General:

The Orient Creek Swamp is a low area lying between Beaver Lake and Orient Creek itself. The Orient Swamp area forms the headwaters of Orient Creek. Many natural uncontaminated springs were observed in this area. A small almost insignificant ridge separates Beaver Lake and the Orient Swamp area.

The only contaminated waste gaining access to this swamp area is the Decant Swamp Overflow (see Section (B) above).

The swamp water itself is coffee-brown in colour and simply reflects normal swamp conditions. A spot check for ferrous and ferric iron indicated no ferrous iron present and only a trace amount of ferric iron.

(3) Chemical Characteristics of the Orient Creek Swamp Runoff:

Iron as Fe		Sulphate	pH	Uranium	Solids		Chloride	Hardness	
Total	Diss.	as SO ₄ ⁼	at Lab.	as U	Total	Susp.	Diss.	as CaCO ₃	
0.0	--	47	4.4	--	130	5	125	4	60

NOTE: All results (except pH) expressed as ppm.

NOTE: The above sample (of Orient Creek) was taken just before Orient Creek wanders onto the "spill" area.

(F) North Dam Seepage

Photograph D

North Dam Seepage

(1) Flow:

Variable but between 30,000 - 50,000 gpd.

(2) Physical Description of the Waste:

The North Dam Seepage Flow is generally light (faint) yellow in colour. The waste is rarely turbid.

(3) Chemical Characteristics of the North Dam Seepage:

	Concentration ppm	
	<u>June 18, 1969</u>	<u>October 21 & 22</u>
Total Iron as Fe	380	3,200
Ferrous Iron		1,750
pH at Lab.	2.2	2.0
Sulphate as $\text{SO}_4^{=}$	11,500	7,440
Uranium as U		7.2
Zinc as Zn		11.4
Nickel as Ni		3.2
Cobalt as Co		3.8
Copper as Cu		3.6
Manganese as Mn		5.6
Titanium as Ti	2.0	
Total Solids	19,900	
Susp. Solids	30	
Diss. Solids	19,870	
Nitrogen (Nitrate)	0.63	
Nitrogen (NH_3)	35.0	
Chloride as Cl^-	less than 1.0	
COD	478	
Acidity as CaCO_3	11,600	

(4) Direction of Flow:

The North Dam Seepage flows through rough country to Quirke Lake. Before it reaches Quirke Lake, the waste flow is augmented by seepage from the "Swallow Hill" area of the tailings mass. Also, the entire overflow from Beaver Lake is directed into this seepage course.

(G) Swallow Hill Seepage

Photograph E

One of the sources of seepage from the Swallow Hill area.

Photograph F

A portion of the Swallow Hill seepage as it makes its way to join the seepage from the North Dam.

(1) Flow:

Variable, generally 10,000 - 20,000 gpd.

(2) Physical Description of the Waste:

Identical in appearance to North Dam Seepage. Deep amber colour in stagnant pools.

(3) Chemical Characteristics of the Swallow Hill Seepage:

<u>Iron as Fe</u> <u>Total</u>	<u>Ferrous</u>	<u>Sulphate</u> <u>as SO₄</u>	<u>pH</u> <u>at Lab.</u>	<u>Uranium</u> <u>as U</u>	<u>Zinc as</u> <u>Zn</u>	<u>Ni</u>	<u>Co</u>
3760	185.0	8,920	1.8	5.2	11.5	5.0	7.6

NOTE: All results (except pH) expressed as ppm.

(4) Direction of Flow:

Seepage from Swallow Hill flows down a wooded ravine to meet the North Dam Seepage in an area leveled by tailings in 1964. From here, the waste flows to Quirke Lake.

(H) Seepage to Beaver Lake

Photograph G

A section of Beaver Lake with the Stanrock tailings area in the back-

ground. Seepage to the Lake occurs along the entire ridge.

(1) Flow:

Not known, erratic, generally during periods of precipitation.

(2) General:

During periods of heavy rain, faint yellow and amber-coloured wastes have been seen running down the steep slope that stretches from the tailings area to the Lake. The Lake intercepts these wastes and overflows (forming Beaver Creek) to a valley that leads to Quirke Lake (see below "Beaver Creek") (also see Section entitled "Deflation - Wind Erosion").

(I) Beaver Creek

(1) Flow:

Variable, generally 100,000 - 200,000 gpd, occasionally increases to 1,000,000 gpd.

(2) Physical Description of the Waste:

The water in Beaver Creek is generally pale-yellow in colour. In stagnant pools, the water becomes yellow-amber in colour.

(3) Chemical Characteristics of Beaver Creek:

Iron as Fe		pH	Uranium	Mn	Zn	Co	Ni
Total	Ferrous	at Lab.	as U				
545	95.00	3.8	1.8	8.44	5.35	0.0	2.56

NOTE: All results (except pH) expressed as ppm.

(4) Direction of Flow:

Beaver Lake overflows to form Beaver Creek. Beaver Creek flows

through a rough valley with a swampy bottom. Eventually, Beaver Creek joins the North Dam Seepage stream (below its junction with the Swallow Hill Seepage stream) and then flows directly to Quirke Lake.

Lakes and Watercourses Influenced Directly by the Stanrock Tailings Area

(1) Cinnabar Creek

Photograph H

Cinnabar Creek as it approaches the mine road.

(a) Source(s)

- (i) Decant Overflow (see Section entitled "Sources of Liquid Wastes" - Part C, Tailings Area - Subsection A) Decant Overflow.

(ii) South Dam Seepage (see Section entitled "Sources of Liquid Wastes" - Part C, Tailings Area - Subsection C) South Dam Seepage.

(iii) Mine Road Seepage (see Section entitled "Sources of Liquid Wastes" - Part C, Tailings Area - Subsection D) Mine Road Seepage.

(b) Chemical Characteristics of Cinnabar Creek

	Concentration ppm	
	<u>June 18, 1969</u>	<u>October 21 & 22</u>
Total Solids	5,900	3,990
Susp. Solids	20	170
Diss. Solids	5,880	3,820
Total Iron as Fe	284	520
Ferrous Iron as Fe		8.25
pH at Lab.	2.5	2.7
Sulphate as $\text{SO}_4^{=}$	3,200	2,200
COD	81	
Acidity as CaCO_3	3,140	
Chloride as Cl^-	trace <1	1.0
Nitrogen (Nitrate)	0.25	
Nitrogen (NH_3)	17.5	
Titanium as Ti	1.0	
Uranium as U		3.2
Zinc as Zn		5.0
Nickel as Ni		1.5
Cobalt as Co		1.8

(c) Direction of Flow

Cinnabar Creek follows the fossil course of the stream that drained the lake that originally occupied the depression now filled by the present tailings body. Cinnabar Creek flows into Orient Creek within half a mile of the Stanrock tailings area. Orient Creek flows into Pink Dragon Lake.

(2) Orient Creek

(a) Source(s)

(i) Orient Creek Swamp (see Section entitled "Sources of Liquid Wastes" - Part C, Tailings Area - Subsection E) Orient Creek Swamp.

(b) General

Orient Creek is initially formed in the vicinity of Beaver Lake. A myriad of springs ensures that the headwaters of this stream are uncontaminated. However, within one mile of the tailings area, Orient Creek is joined by Cinnabar Creek. The union of one with the other results in mutual destruction with regard to water quality. The combined unit (still known as Orient Creek) discharges into Pink Dragon Lake.

(3) Beaver Creek

(see Section entitled "Sources of Liquid Wastes" - Part C, Tailings Area - Subsection I) Beaver Creek.

(4) Beaver Lake

Photograph I

Beaver Lake looking towards the Orient Creek Swamp.

Photograph J

The swampy end of Beaver Lake. A ridge about 2 feet high (on which the observer is standing) separates Beaver Lake and the Orient Creek Swamp.

(a) General

Beaver Lake lies between the Stanrock tailings area and Quirke Lake. Its geographical position is so delicately balanced that it could be easily forced to flow into the Lower Serpent River basin. However, as far as it is known, Beaver Lake has always discharged to Quirke Lake (via Beaver Creek) (Beaver Creek is approximately 3500 feet in length).

The surface of Beaver Lake lies 61 feet below the surface of the

tailings area but at the same time lies 157 feet above the surface of Quirke Lake.

1300 feet in length and 400 feet in width, Beaver Lake occupies an area of less than 12 acres. Its average depth is assumed to be 10 feet.

When viewed in depth, Beaver Lake is amber-brown in colour.

(b) Chemical Characteristics of Beaver Lake

	Concentration ppm	
	<u>June 18, 1969</u>	<u>October 21 & 22</u>
Total Solids	1320	
Susp. Solids	15	
Diss. Solids	1305	
Total Iron as Fe	56	66.5
Ferrous Iron as Fe		20.0
Sulphate as SO_4^{--}	660	
Nitrogen (Nitrate)	0.53	
Nitrogen (NH_3)	2.0	
pH at Lab.	2.7	2.8
COD	4	
Acidity as CaCO_3	820	
Chloride as Cl^-	trace <1	
Titanium as Ti	0.00	
Manganese as Mn		2.12
Zinc as Zn		1.15
Uranium as U	0.34	0.3
Cobalt as Co		1.7
Nickel as Ni		1.20

(5) Pink Dragon Lake

Photograph K

The inlet to Pink Dragon Lake. Tails, carried from the major "spill" area by Orient Creek, have formed a small deltaic deposit at the inlet. The very dark amber-yellow colour of the water in the background should be noted.

Photograph L

Pink Dragon Lake. The inlet is clearly visible. Note the "South Dam" of the tailings area and the Stanrock mill in the background.

Photograph M

Pink Dragon Lake (in the foreground) is separated from an arm of Quirke Lake (in the background) by a narrow ridge. This photograph shows only the lower (discharge) end of Pink Dragon Lake. Half Moon Lake lies .5 miles downstream.

Photograph N

The discharge end of Pink Dragon Lake. A large abandoned beaver dam helps to increase the retention time of the area.

(a) General

Pink Dragon Lake is much larger than Beaver Lake. Its main source of water is Orient Creek (which at this point has augmented its flow by union with Cinnabar Creek). Several natural springs also discharge into Orient Creek upstream of Pink Dragon Lake but, before they do, these microstreams first wander over the sulphate and acid saturated ground of the main "spill" area. By the time they reach Orient Creek, their chemical characteristics are identical to those of the

seepage streams arising within the main tailings mass.

(b) Pink Dragon Lake Discharge Flow Rates

The flow rate in the stream (an extension of Orient Creek) that carries the discharge from Pink Dragon Lake to Half Moon Lake is fairly accurately known. The average flow is 1,525,000 imperial gallons per day (24 hours). A minimum flow of 221,000 gpd has been recorded. The maximum flow recorded was 14,688,000 gpd.

(c) Chemical Characteristics of the Pink Dragon Lake Overflow

October 29, 1969

<u>Total</u>	<u>Solids</u>		<u>pH</u>	<u>Iron as Fe</u>		<u>Sulphate as SO₄⁼</u>	<u>Chlorides as Cl⁻</u>	<u>Uranium as U</u>
	<u>Susp.</u>	<u>Diss.</u>		<u>Total</u>	<u>Diss.</u>			
1610	50	1560	2.9	166	120	900	19	0.5

NOTE: All results (except pH) expressed as ppm.

NOTE: The overflow from Pink Dragon Lake runs quickly and directly to Half Moon Lake. The watercourse is enclosed by a narrow steep-walled valley. The stream flow is not augmented during its journey by major flows from any other source.

(6) Half Moon Lake

Photograph 0

The upper end of Half Moon Lake. The overflow from Pink Dragon Lake enters at the extreme left of the photograph. Uncontaminated water from another lake in the area enters Half Moon Lake through the small bay in the upper central portion of the photograph.

(a) General

Half Moon Lake receives the discharge from Pink Dragon Lake. In turn, Half Moon Lake overflows to May Lake.

(c) Chemical Characteristics of Half Moon Lake

	Concentration ppm <u>February 1968</u>
Total Solids	2410
Susp. Solids	6
Nitrogen (NH ₃)	19.7
Nitrogen (Total Kjeldahl)	25.0
Nitrogen (Nitrite)	--
Nitrogen (Nitrate)	0.50
Total Phosphorus as PO ₄	0.52
Sol. Phosphorus as PO ₄	0.12
Hardness as CaCO ₃	640
Acidity as CaCO ₃	796
Chloride as Cl ⁻	53
Sulphate as SO ₄ ⁼	1140
pH at Lab.	3.0
Total Iron as Fe	285
Manganese as Mn	0.94

SOURCES OF SOLID WASTES

(a) Conventional Mining and Milling Operations

Conventional mining and milling operations in the Elliot Lake district have produced enormous quantities of finely divided waste material known in the industry as "tailings". For all intents and purposes, a ton of ore delivered to any operating mill in the Elliot Lake district will ultimately result in a ton (dry) of tailings being discharged to a holding basin. Generally speaking, the holding basins are located in the vicinity of the mill and are designed to receive and hold

tailings for the entire producing life of the mine.

The sole function of a tailings area is to provide enough retention time so that a complete separation of solids and liquids can occur. Under certain circumstances, the natural separation of solids and liquids can be enhanced by the addition of specific chemical compounds. The material settling within a tailings area, if inert, and if the retaining structures are in good condition, will remain within the tailings area. The liquid phase, on the other hand, generally leaves the tailings area via an overflow structure called the "decant". Varying quantities of the liquid also escape the tailings area as seepage.

During the years 1959 to 1963, the Stanrock mill processed over 5,130,000 tons of ore. In addition, the combined production for the years 1958 and 1964 probably exceeded $1\frac{1}{4}$ million tons. Since the tailings were always discharged to the tailings area as a slurry (a mixture of rock particles and water), a great deal of water was associated with this production. At full capacity, the daily discharge to the tailings area was approximately 3,000 tons of solids (52 per cent minus 200 mesh) and 8,000 tons (1,600,000 imperial gallons) of water.

(b) Bacterial Leaching

In general, the bacterial leaching operations being carried out at the Stanrock mine do not generate solid wastes. Small amounts of iron sludge may accumulate in some of the tanks but the disposal of this material is not considered to be a problem.

CHARACTERISTICS OF THE STANROCK TAILINGS AREA

(a) Size

The main body of the Stanrock tailings area occupies approximately 130 acres (see Sheets #1 and/or #2). A secondary tailings area (see Photograph A2) of 10 to 20 acres in extent lies immediately downstream from the main tailings body. This secondary tailings area was created accidentally on June 9, 1964, when the decant structure on the South Dam failed.

NOTE: In various sections of this report, this secondary tailings area is referred to as the major "spill" area.

Photograph A2

The major spill area.

A minor 1 to 2 acre site near the Quirke Lake end of the North Dam was covered with tailings on April 17, 1964 (see Photograph V) when a dyke collapsed.

(b) Depth of Tailings

The actual average depth of tailings in the main tailings area is not known by the Commission. However, it seems safe to assume that the average mean depth is at least 30 feet. The tails are almost level with the top of the South Dam which is reported to have a height of 75 feet.

(c) Precipitation

The average total annual precipitation in the Blind River (Elliot Lake) area is 32.58 inches.

STABILITY OF THE STANROCK TAILINGS AREA

(a) North Dam

Photograph P

The North Dam (looking towards Quirke Lake).

The North Dam appears to be the most stable of all the retaining structures on the property. With the exception of the extreme Quirke Lake end of the dam (which has been somewhat dissected by erosion), there is no visual evidence of structural deterioration. Seepage occurs along the rocky toe of the dam but this is to be expected.

(b) Swallow Hill Area

The term "Swallow Hill" in this report is applied to the area of the Stanrock tailings mass that directly overlooks Beaver Lake. In places along this

ridge, the tails have sheared away to produce a vertical face. The base of this cliff is one of the sources of the Swallow Hill Seepage. The cliff itself is stable enough to support a colony of swallows on a year-to-year basis but is so unstable that the field staff was able to dislodge huge semi-coherent masses of tailings with very little effort.

Tailings from this area are ultimately finding their way into the stream leading to Quirke Lake.

(c) South Dam

Photograph Q

The massive South Dam. A small portion of the decant seepage can be seen in the foreground.

The South Dam, reportedly 75 feet high, is the most important single manmade structure on the property. It is primarily this dam that is responsible for retaining the tails in the basin.

Viewed from a distance, the South Dam appears to be a stable, well-formed reliable unit. The downstream face of the dam, however, is periodically coated with a thin film of wind-blown sand and it is this opaque snow-white layer that hides the many flaws in the structure from the casual viewer.

The item of major concern at the present time is the rapidly progressing deterioration of the decant structure that penetrates the South Dam itself. The concrete tunnel that serves to carry liquid waste from the pond behind the top of the dam to the watercourse at the foot of the dam has sometime in the immediate past disintegrated at one or two points along its length.

Tails, overlying the break, have run, under the influence of gravity, into the decant chamber and have been washed out into Cinnabar Creek. As this process continued, a large crater above the break was formed.

Photograph S

The crater above the concrete tunnel at the Stanrock decant.

At the present time, the crater is enlarging itself. While the process mechanism is easily understood, it is most apparent during a heavy rainstorm when Cinnabar Creek is forced to carry a heavy load of suspended solids.

Caving is also occurring on the downstream face of the South Dam.

Photograph T

Caving on the Face of the South Dam.

The caved material is generally washed into the watersheds of Cinnabar and Orient Creeks.

An intricate shear structure has developed between the area of the decant pond and the face of the South Dam itself. This extensive system of cracks, some of which are 2 or 3 inches in width, extend to an undetermined depth into the tailings area. One minor crack has been seen that extended to the surface of the saturated unconsolidated tailings only 4 feet below the surface of the tailings area. The larger cracks seem to radiate from the decant pond itself and come very close to penetrating the face of the South Dam. Undoubtedly, these cracks are

deeper and may extend to the saturated zone near the dam face. The cracks (which tend to become filled with coarse drifting sand at the surface of the tailings area) should be regarded as a definite threat to the stability of the area. The cracks may have originally formed as a result of the desiccation and subsequent slumping of the beds of tailings near and in the decant pond as the water level was lowered.

DEFLATION (WIND EROSION) ON THE STANROCK TAILINGS AREA

Photograph U

The Erosive Action of Wind at Stanrock.

Deflation is by far the most important work of the wind, and its significance can scarcely be over-estimated. The lifting of the material from the

surface is largely the work of eddies and irregularities of movement of the wind, including many conflicting cross-currents.

The most important are, of course, the whirling eddies of whirlwinds and tornadoes but many minor currents, due in part at least to the irregularities of the surface, are active in lifting the dust and fine sand. The ordinary convection currents of the atmosphere carry this finer material up to great heights. The force, which moves the particle, is due to the direct impact of the wind, plus the friction along its surface. The force of the direct impact varies with the velocity of the wind, and for a given velocity with the cross-section of the particle in the plane perpendicular to the direction of the wind, as well as with the orientation of the particle, and it is of course most efficient when a smooth plane is opposed to the wind. The resistance of the particle varies with its mass, i.e., its size and specific gravity, and to some extent, its form. In general, the velocity of the wind necessary to carry a spherical particle of given specific gravity varies as the square of the radius, and conversely, the radius of a particle which can be supported by wind varies as the square root of the velocity. It was found that a uniform upward current of air will keep suspended quartz grains, the size of which varied with the velocity, as follows:

<u>Velocity in meters per second</u>	<u>Diameter of quartz grain in mm.</u>
0.50	0.04
1.00	0.08
2.00	0.16
3.00	0.25
4.30	0.35
5.00	0.41
6.00	0.49
7.00	0.57
8.00	0.65
9.00	0.73
10.00	0.81
11.00	0.89
12.00	0.97
13.00	1.05

From this, the formula $V = Kr$ was deduced, where K is a constant for the conditions of experiment and r the radius of the particle.

As a result of numerous measurements, it was concluded that the average largest size of quartz particles that can be sustained in the air by ordinary strong winds is about 0.1 mm. in diameter but the largest particle that can be

transported (not held in suspension) is nearer 2 mm. in diameter, while gravel, the size of peas, may in rare cases be carried along by the wind. This, however, represents the limit of ordinary deflation.

The following table gives the approximate size of quartz sand grains moved by varying wind velocities.

<u>Strength of wind in meters per second</u>	<u>Maximum diameter of sand grains in mm.</u>
4.5 - 6.7	0.25
6.7 - 8.4	0.5
9.8 - 11.4	1.0
11.4 - 13.0	1.5

These results are only approximate.

Problems due to wind-blown tailings have been encountered at nearly every mining operation in the Elliot Lake district. The situation at Stanrock is very typical. A large unbroken expanse of tailings is exposed to the action of the wind. The direction of the wind is of little consequence.

The attitude of mining management with regard to wind erosion in the Elliot Lake district may be typified by a comment taken from our Stanrock file in Toronto. The following sentence formed part of a letter sent to the Commission by a former manager at the Stanrock property:

"I can't think of any place where blown sand could bother fewer people".

Up until the 1968 field season, the staff of the Ontario Water Resources

Commission generally supported this statement. However, during the summer of 1968, it became apparent that swampy areas and "pot-hole" lakes located in the vicinity of the tailings areas were contaminated although no normal link to the tailings areas could be established. The colour changes in these waters were observed to be identical to the colour changes taking place in waters that were directly exposed to the uranium tailings, and therefore, the possibility of the aerial transport of finely divided tailings from the tailings areas to the waters in question could not be ignored.

The first step in our preliminary investigation of the airborne tailings was to secure a sample of tailings that we knew had already undergone some degree of aerial transport. Since deposits of wind-blown sand were easily recognizable and readily available in and around the Stanrock tailings area, we selected this as our sample source.

The following are the actual average analytical results of the samples in question:

Total Iron as Fe	2.05%
Calcium as Ca	0.013%
Sodium as Na	0.006%
Potassium as K	0.316%
Phosphorus as P	0.033%
Aluminum as Al	0.68%
Titanium as Ti	0.046%

The above values were obtained by leaching the sample and therefore represent the material available to the natural processes of chemical decomposition and reaction. The iron content is, quite unexpectedly, very high. The bulk of

the sample, of course, was composed of silicon dioxide (an "inert" material in this context).

A sieve analysis was also run on the wind-blown samples and serves, at least, to identify the size fractions that can be moved by the wind in the Elliot Lake area.

Retained by No. 10 mesh	0.021%
Passed by No. 10, Retained by No. 25 mesh	0.029%
Passed by No. 25, Retained by No. 45 mesh	1.47%
Passed by No. 45, Retained by No. 100 mesh	85.3%
Passed by No. 100, Retained by No. 200 mesh	11.7%
Passed by No. 200 mesh	1.44%

Assuming that the samples were representative, it is immediately apparent that particles that will not pass through a 45 mesh screen (U.S. Standard Sieve) are generally too large to be subject to aerial transportation in the Elliot Lake area. On the other hand, particles that are small enough to pass through a 45 mesh screen will undergo active aerial transportation should the required conditions present themselves.

It is necessary to point out that the above sieve analysis reflects the characteristics of the wind-blown deposits after they have undergone some degree of aerial transport. In other words, we have to assume that most of the fine (-200 mesh) material was removed by wind action from the deposits before we sampled them.

Analysis of the residual -200 mesh fraction revealed that it contained 15.2 per cent iron as Fe. In other words, the fine dust that is blown from the tailings area and that eventually settles on the land, and in the stagnant lakes

and streams in the immediate area, could possibly contain up to 30 per cent sulphides.

Sulphides are the prime sources of chemical contamination in the Elliot Lake district.

SAMPLING AND ANALYSIS

(a) Weather Conditions During Sampling Periods

(i)	June 18, 1969	clear, hot, dry
(ii)	October 20, 1969	cloudy, some rain, cool
(iii)	October 21, 1969	cloudy periods, no rain, cold, some light snow towards evening
(iv)	October 22, 1969	clear and cold, windy, tailings area covered with 1 inch of light dry snow
(v)	November 6, 1969	partly cloudy, no pre- cipitation, ground bare

(b) Sampling

Routine precautions were followed throughout the entire sampling programme. All equipment was checked for cleanliness before each sample was taken. All chemical samples were put in separate clean glass containers. The containers, in groups of eight, were shipped via railroad to our Laboratory in Toronto.

(c) Analysis

All chemical analyses were performed by skilled technicians using the most modern equipment available. All results have been cross-checked.

The analytical results quoted in this report (with the exception of pH) are expressed as parts per million (ppm) unless otherwise indicated. pH is expressed in the standard units of the pH scale. 1 part per million is approximately equivalent to 1 milligram (mgm.) per litre or 1 pound per 100,000 imperial gallons.

WASTE LOADINGS (GROSS)

The waste loadings calculated in this Section are based on the chemical results available (most of which are very accurate) and on the average flows observed for each "stream".

Part A - Waste Loadings to Quirke Lake

(a) North Dam Seepage

Flow - 40,000 gpd

	Concentration ppm	Loading lbs./day
Total Iron as Fe	3,200	1,280
Ferrous Iron	1,750	700
Sulphate as $\text{SO}_4^{=}$	7,440	2,976
Total Solids	19,900	7,960
Susp. Solids	30	12
Diss. Solids	19,870	7,948
Nitrogen (Nitrate)	0.63	.3
Nitrogen (NH_3)	35.0	14
Chloride as Cl^-	less than 1.0	less than .4
COD	478	191
Uranium As U	7.2	2.9
Zinc as Zn	11.4	4.6
Nickel as Ni	3.2	1.3
Cobalt as Co	3.8	1.5
Copper as Cu	3.6	1.4
Manganese as Mn	5.6	2.2
Titanium as Ti	2.0	.8

(b) Swallow Hill Seepage

Flow - 15,000 gpd

	Concentration ppm	Loading lbs./day
Total Iron as Fe	3,760	564
Ferrous Iron	185	28
Sulphate as $\text{SO}_4^{=}$	8,920	1,338
Uranium as U	5.2	.8
Zinc as Zn	11.5	1.7
Nickel as Ni	5.0	.8
Cobalt as Co	7.6	1.1

(c) Beaver Creek

Flow - 150,000 gpd

	Concentration ppm	Loading lbs./day
Total Iron as Fe	545	818
Ferrous Iron	95	143
Sulphate as $\text{SO}_4^{=}$	660	990
Uranium as U	1.8	2.7
Zinc as Zn	5.35	8.0
Nickel as Ni	2.56	3.8
Cobalt as Co	0.0	--
Manganese as Mn	8.44	12.7

The total gross loadings to Quirke Lake (excluding the compressor cooling water) are as follows:

	Loading lbs./day
Total Iron as Fe	2,662
Ferrous Iron	871
Sulphate as $\text{SO}_4^{=}$	5,304
Uranium as U	6.4
Zinc as Zn	14.3
Nickel as Ni	5.9
Cobalt as Co	2.6
Manganese as Mn	14.9 - this value is a minimum

Part B - Waste Loadings to the Lower Serpent River System

(a) Decant Overflow

Flow - 150,000 gpd

	Concentration ppm	Loading lbs./day
Total Iron as Fe	368	552
Ferrous Iron	156	234
Sulphate as $\text{SO}_4^{=}$	6,900	10,350
Total Solids	13,460	20,190
Susp. Solids	25	38
Diss. Solids	13,435	20,152
Nitrogen (Nitrate)	10.0	15
Nitrogen (NH_3)	100	150
Chloride as Cl^-	97	145
COD	278	417
Uranium as U	3.2	4.8
Zinc as Zn	9.4	14.1
Nickel as Ni	1.5	2.3
Cobalt as Co	1.5	2.3
Manganese as Mn	3.6	5.4
Copper as Cu	2.2	3.3
Titanium as Ti	3.6	5.4

(b) South Dam Seepage

Flow - 50,000 gpd

	Concentration ppm	Loading lbs./day
Total Iron as Fe	1,530	765
Ferrous Iron	50	25
Sulphate as $\text{SO}_4^{=}$	6,280	3,140
Uranium as U	7.4	3.7
Zinc as Zn	10	5
Nickel as Ni	2.3	1.2
Cobalt as Co	3.1	1.6
Copper as Cu	2.1	1.1

(c) Mine Road Seepage

Flow - 30,000 gpd

	Concentration ppm	Loading lbs./day
Total Iron as Fe	120	36
Ferrous Iron	3.5	1
Sulphate as $\text{SO}_4^{=}$	760	228
Uranium as U	0.4	.1
Zinc as Zn	0.98	.3
Nickel as Ni	0.45	.1
Cobalt as Co	0.40	.1
Copper as Cu	0.26	.1

(d) Orient Creek Swamp Runoff

Flow - 100,000 gpd

The headwaters of Orient Creek, at present, contribute 47 pounds of sulphates per day to the Pink Dragon Lake area. Most of the sulphate content is derived from the decant swamp overflow, etc.

The total gross loadings from the immediate tailings area to the Lower Serpent River System are the following:

	Loading lbs./day
Total Iron as Fe	1,353
Ferrous Iron	260
Sulphate as $\text{SO}_4^{=}$	13,765
Uranium as U	9
Zinc as Zn	19
Nickel as Ni	4
Cobalt as Co	4
Copper as Cu	5

Pink Dragon Lake provides some retention time for the acidic wastes. Therefore, the ferrous iron tends to oxidize and settle. Pink Dragon Lake overflows to Half Moon Lake. The rate of overflow is accurately known and averages 1,525,000 imperial gallons per day.

Waste Loadings - Pink Dragon Lake Overflow

	Concentration ppm	Loading lbs./day
Total Solids	1,610	24,553
Susp. Solids	50	763
Diss. Solids	1,560	23,790
Total Iron as Fe	74	1,129
Diss. Iron	54	824
Sulphate as $\text{SO}_4^{=}$	900	13,725
Chloride as Cl^-	19	290
Uranium as U	0.5	7.6

Total Waste Loadings to the Serpent River Watershed
From Stanrock Uranium Mines Limited

	Loading lbs./day
Total Iron as Fe	4,015
Ferrous Iron	1,131
Sulphate as $\text{SO}_4^{=}$	19,069
Uranium as U	15
Zinc as Zn	33
Nickel as Ni	10
Cobalt as Co	7
Copper as Cu	5 * minimum value - should be doubled
Manganese as Mn	15 * minimum value - should be doubled

RADIOLOGICAL DATA

pC_i/litre = picocuries per litre

The radiological characteristics of the various watercourses and waste streams in the vicinity of Stanrock Uranium Mines Limited are as follows:

(a) Quirke Lake Intake Water

This is the "fresh" water pumped from Quirke Lake. It is only used for compressor cooling and for process make-up in the mill.

Radium-226 (Dissolved)	18 pC_i/litre
Gross alpha (Dissolved)	56 pC_i/litre
Gross alpha (Undissolved)	25 pC_i/litre
Gross beta (Dissolved)	75 pC_i/litre
Gross beta (Undissolved)	70 pC_i/litre
Pb-210	--
Th-227	--
Th-228	2 pC_i/litre (filtrate)
Th-230	3 pC_i/litre (filtrate)
Th-232	1 pC_i/litre (filtrate)

(b) Mine Water to Mill

This is the uranium-bearing water that is pumped from underground (the mine) to the ion-exchange circuit (the mill).

Radium-226 (Dissolved)	7 pC_i/litre
Gross alpha (Dissolved)	31,000 pC_i/litre
Gross alpha (Undissolved)	3,500 pC_i/litre
Gross beta (Dissolved)	23,000 pC_i/litre
Gross beta (Undissolved)	2,000 pC_i/litre
Pb-210	--
Th-227	--
Th-228	1,200 pC_i/litre (filtrate)
Th-230	12,000 pC_i/litre (filtrate)
Th-232	3,100 pC_i/litre (filtrate)

(c) Barren to Mine

This is the "effluent" from ion exchange. Theoretically stripped of its uranium content, the "barren" is generally returned to the mine.

Radium-226 (Dissolved)	103 pCi/litre
Gross alpha (Dissolved)	23,000 pCi/litre
Gross alpha (Undissolved)	120 pCi/litre
Gross beta (Dissolved)	22,000 pCi/litre
Gross beta (Undissolved)	100 pCi/litre
Pb-210	--
Th-227	--
Th-228	2,600 pCi/litre (filtrate)
Th-230	33,000 pCi/litre (filtrate)
Th-232	6,700 pCi/litre (filtrate)

(d) North Dam Seepage

Radium-226 (Dissolved)	12 pCi/litre
Gross alpha (Dissolved)	31,000 pCi/litre
Gross alpha (Undissolved)	300 pCi/litre
Gross beta (Dissolved)	13,000 pCi/litre
Gross beta (Undissolved)	310 pCi/litre
Pb-210	--
Th-227	--
Th-228	850 pCi/litre (filtrate)
Th-230	7,500 pCi/litre (filtrate)
Th-232	1,800 pCi/litre (filtrate)

(e) Tailings Area Decant

Radium-226 (Dissolved)	15 pCi/litre
Gross alpha (Dissolved)	13,000 pCi/litre
Gross alpha (Undissolved)	400 pCi/litre
Gross beta (Dissolved)	8,500 pCi/litre
Gross beta (Undissolved)	220 pCi/litre
Pb-210	--
Th-227	--
Th-228	420 pCi/litre (filtrate)
Th-230	5,900 pCi/litre (filtrate)
Th-232	850 pCi/litre (filtrate)

(f) Influent to "Spill" Area (i.e., Cinnabar Creek at Mine Road)

Radium-226 (Dissolved)	1.5 pCi/litre
Gross alpha (Dissolved)	3,100 pCi/litre
Gross alpha (Undissolved)	420 pCi/litre
Gross beta (Dissolved)	2,400 pCi/litre
Gross beta (Undissolved)	560 pCi/litre
Pb-210	--
Th-227	--
Th-228	87 pCi/litre (filtrate)
Th-230	1,200 pCi/litre (filtrate)
Th-232	180 pCi/litre (filtrate)

(g) Effluent from "Spill" Area (i.e., Orient Creek Downstream of the Orient Creek-Cinnabar Creek Junction but before Pink Dragon Lake)

Radium-226 (Dissolved)	less than 1 pCi/litre
Gross alpha (Dissolved)	2,400 pCi/litre
Gross alpha (Undissolved)	140 pCi/litre
Gross beta (Dissolved)	2,100 pCi/litre
Gross beta (Undissolved)	140 pCi/litre
Pb-210	--
Th-227	--
Th-228	65 pCi/litre (filtrate)
Th-230	820 pCi/litre (filtrate)
Th-232	130 pCi/litre (filtrate)

(h) Beaver Lake Overflow (i.e., Beaver Creek)

Radium-226 (Dissolved)	8 pCi/litre
Gross alpha (Dissolved)	590 pCi/litre
Gross alpha (Undissolved)	36 pCi/litre
Gross beta (Dissolved)	520 pCi/litre
Gross beta (Undissolved)	65 pCi/litre
Pb-210	--
Th-227	--
Th-228	35 pCi/litre (filtrate)
Th-230	530 pCi/litre (filtrate)
Th-232	85 pCi/litre (filtrate)

(i) Combined North Dam Seepage-Swallow Hill Seepage

NOTE: In this instance, the North Dam Seepage has been neutralized.

Radium-226 (Dissolved)	1 pCi/litre
Gross alpha (Dissolved)	3,700 pCi/litre
Gross alpha (Undissolved)	320 pCi/litre
Gross beta (Dissolved)	1,800 pCi/litre
Gross beta (Undissolved)	220 pCi/litre
Pb-210	--
Th-227	--
Th-228	110 pCi/litre (filtrate)
Th-230	1,600 pCi/litre (filtrate)
Th-232	270 pCi/litre (filtrate)

(j) Beaver Creek at Quirke Lake (with Neutralization)

Radium-226 (Dissolved)	less than 1 pCi/litre
Gross alpha (Dissolved)	7,770 pCi/litre
Gross alpha (Undissolved)	1,000 pCi/litre
Gross beta (Dissolved)	5,500 pCi/litre
Gross beta (Undissolved)	840 pCi/litre
Pb-210	--
Th-227	--
Th-228	250 pCi/litre (filtrate)
Th-230	3,800 pCi/litre (filtrate)
Th-232	730 pCi/litre (filtrate)

The following paper entitled "MPC (Maximum Permissible Concentration)

Objectives for Drinking Water Contaminated by Certain Uranium-Thorium Daughter Mixtures" was written by L. B. Leppard of the Industrial Hygiene Branch of the Ontario Department of Health. The paper has been included unchanged as a part of this report because it refers directly to the radiological problems in the Elliot Lake district:

"Background

The milling of uranium ores results in the accumulation of large tonnages of waste material: the appropriate figure in the case of the Elliot Lake mining area in Ontario was something over fifty million tons at the end of 1964. This represents the mill discharge from eleven mining enterprises operating on the average for considerably less than ten years, since only three of the original eleven remain active today.

The waste, in which the residual sulphuric acid from the leaching process has been neutralized with lime, is pumped as a slurry to settling lagoons and from these there ensues a relatively clear overflow into the streams and lakes of the surrounding drainage area.

For our purpose here today the waste may be described as a low specific activity material. The mill recovery process loses very little uranium, but it does reject the radioactive daughters of uranium, practically in toto, to tails. Since these daughters are either very long lived, or if short lived, are usually capable of being regenerated from long lived precursors, one would have been entitled to wonder, ten years ago, whether persistent radioactive contamination of the receiving waters would not be inevitable; and this in spite of all efforts to insolubilize the tailings before they were deposited.

In advance there was no way of knowing for sure. A priori, it might look as if twenty-nine pounds of the insoluble sulphate of Radium-226, well and truly dispersed throughout fifty million tons of solids (that is, at a specific activity of 300 picocuries per gram), would not leak away fast enough to matter at some more or less distant water intake. After all, 300 picocuries per gram is a figure only about 100 times higher than that representing the average occurrence of radium in the earth's crust and it is less than 0.002 microcuries per gram, a figure frequently used in foreign statutes (but not in the Atomic Energy Control Act and Regulations of Canada) to mark the lower limit, below which a substance is not legally radioactive. Actually, the problem was too complex for analysis. Detailed information was lacking (and still is) on the chemical form and solubility of the radioactive inclusions in the waste, on the possible modifying effects on solubility exerted by foreign ions, and on water flows and dilution factors.

It was therefore more out of curiosity than anything else that we, in the Ontario Department of Health, began a water-sampling programme in both the Elliot Lake and Bancroft uranium mining areas early in 1958. The only radio-chemical analyses we could manage at the time were for Radium-226 (by the emanation method) and for uranium; and sample handling capacity was low until our new laboratory came into operation in 1961. As it turned out, both uranium and radium showed up regularly in the analyses. The former has never attained levels of any

significance in drinking water, but the latter, while at first giving no cause for concern, seemed well worth watching as the most important radio-toxin in the environment. It was, of course, necessary to process a large number of samples from a wide variety of locations before deciding that we have evidence of something more than a transient condition. To date, more than 2000 samples of drinking waters and mill effluents have been put through.

In 1960, during the early part of this work, the 1959 Report of the International Commission on Radiological Protection appeared with its revised mpc's and its new emphasis on population exposure. A review of our radium results on drinking waters in the light of the new mpc's left us with a sense of unease - alarm, or even concern, would be too strong a word - because now many of them were over the line. We were morally certain, at levels of 30 or 40 picocuries of Radium-226 per litre, as they existed in some of the area water supplies, that no one would experience ill effects even after a life-time's consumption. On the other hand, there must obviously be a reluctance on the part of the provincial government to permit indefinite continuation of radium levels in public water supplies a few times higher than an internationally accepted standard. As an interim measure therefore the Ontario Water Resources Commission ordered the affected plants to switch their intakes over to lakes known not to be polluted. As a result no one in these areas, as far as we are aware, is now regularly supplied with domestic water containing more than about 6 picocuries of Radium-226 per litre.

What the future may hold for the polluted waters of the two mining areas is however not our theme today. In any case, that matter is still sub judice and we must await the report of the interdepartmental deputy ministers' committee in whose hands it lies. Our preamble serves merely to provide a natural introduction to the technical problem to which we now turn.

The Problem

Hitherto, the concentration of Radium-226 has been used as the sole index of (radioactive) quality for drinking-water in the uranium mining areas, in disregard of other natural radioisotopes known also to be present. This procedure has been followed in other countries as well and has been justified on the grounds that the toxicity of Radium-226 is an order of magnitude greater than that of any of the rest. If, however, the concentrations of the others are high enough they will have to be taken into account in spite of lower toxicity. Before committing ourselves to complete reliance on the radium index it is therefore desirable to assess the situation in more detail. This would be simple enough if we had representative radioisotopic impurity analyses on the waters in question, but we have not. Such extended analyses tend to be difficult and time-consuming, and although

more attention is being given to them now we have not had staff or facilities to deal with them in the past. Even in the absence of analyses however, some rather infirm inferences are latent in our basic information, so we will proceed to develop these as far as we can.

The Mixture Rule

The I.C.R.P. and other authorities quote "maximum permissible concentrations (mpc's)" of the various radioisotopes on the assumption that each isotope is acting alone, i.e., that there is no exposure due to others.

Thus, if only one radioisotope is present in water, the quality is satisfactory when the actual concentration of the isotope is less than or almost equal to the quoted mpc. This condition can be written

$$C_a \leq C_m$$
$$\text{or } \frac{C_a}{C_m} \leq 1$$

where C_a is the actual concentration and C_m is the quoted mpc.

If however several radioisotopes are present which act selectively on the same critical organ the restrictive condition (known as "the mixture rule") is written as follows

$$\frac{C_{1a}}{C_{1m}} + \frac{C_{2a}}{C_{2m}} + \frac{C_{3a}}{C_{3m}} + \dots \leq 1$$

where the subscripts "1a" and "1m" identify the actual and quoted maximum permissible concentrations of the first radioisotope, and so on for the others.

It will be noted that the terms on the left hand side of this inequality must be non-negative proper fractions. The denominators are text-book constants, and the numerators are the variables. The best the formula can do is to yield the maximum value of one numerator, say C_{1a} , when the others are assigned compatible values. For operating with the formula, a choice has to be made as to which set of denominators to use. For purposes of illustration, we have selected the ICRP (1959) population values.

The stipulation that all the isotopes considered should act selectively on the same organ is important, since otherwise the insults might be regarded as insufficiently related to be additive, so that our mixture rule would not apply. For the present purpose bone will be taken as the critical organ, because many of the uranium and thorium family members are bone-seekers, and bone storage tends to be a long-term proposition.

The Radioisotopes of Interest

Table I lists seven bone-seekers from among the members of the uranium and thorium families. They have been selected on the basis of relatively low mpc under the grouping "soluble, critical organ - bone" in the ICRP tables, and of either substantial half-life or the possibility of regeneration from a copious precursor. Not all are alpha emitters. There are a few other radioisotopes of marginal interest but these would have added little to the case.

TABLE I

SELECTED BONE-SEEKERS FROM THE URANIUM AND THORIUM FAMILIES

Name	Radioisotope	Half-Life Years	Immediate Precursor		Family Ancestor
			Name	Half-Life Years	
Thorium-230		$8.0 \cdot 10^4$	*U-234	$2.48 \cdot 10^5$	Uranium-238
Radium-226		$1.62 \cdot 10^3$	Th-230	$8.0 \cdot 10^4$	" "
Lead-210		19.4	**Ra-226	$1.62 \cdot 10^3$	" "
Protactinium-231		$3.43 \cdot 10^4$	***Th-231	(25.6 hours)	Uranium-235
Actinium-227		21.6	Pa-231	$3.43 \cdot 10^4$	" "
Thorium-232		$1.39 \cdot 10^{10}$	--	--	Thorium-232
Radium-228		6.7	Th-232	$1.39 \cdot 10^{10}$	" "

*The U-234 is removed in the uranium recovery process, leaving no effective precursor.

**Ra-226 is not the immediate precursor of Pb-210. A decay chain of five short-lived members intervenes. One of these is the gas Radon-222. To the extent that it escapes, successor Pb-210 is lost to the environment.

***Th-231 decays away quickly, leaving no effective precursor once the ancestor U-235 has been removed.

The Raw Material

Since mpc's are expressed in terms of activity per unit volume, we must first gain some idea of both the absolute and relative curie quantities of the radioisotopes with which we may have to deal. The calculation can most conveniently be related to the Elliot Lake area, where up to the end of 1964 approximately 51,000 tons of U_3O_8 , or 42,000 tons of uranium element, had been mined and milled. Of these 42,000 tons, 99.3% or 41,700 tons are Uranium-238 and 0.7%, or 300 tons, are Uranium-235. Further, according to analyses published by Rio Algom Limited, the thorium to uranium ratio in the ore is about 1 to 4, so that some 10,500 tons of thorium element have also been brought up out of the ground. In terms of curies these tonnages are:

Uranium-238 12,600 curies

Uranium-235 585 curies

Thorium-232 1,050 curies

It seems realistic to assume that the uranium and thorium daughters are present in the ore at equilibrium activity. A question might be raised about Lead-210, since it is known that in the process of mining there occurs a seepage escape of precursor Radon-222 into the mine atmosphere. A calculation, based on average volumes and radon content of mine ventilation exhausts, is however reassuring on this point: only a negligible percentage of the equilibrium Lead-210 is lost from the ore in this way.

So it also seems realistic to assume that the uranium daughters and the thorium family are rejected to waste with approximately the activities they possessed in the ore.* We must then ask about the accumulated activities in the waste dump. It seems clear that if the impoundment system retains all, or nearly all the solid material fed to it, each cumulative daughter activity should at any time just about equal the activity of the total amount of ancestor mined up to that time. This is because long lived daughters will have decayed to only a minor extent and because, as can be seen from Table I, the short lived are in the presence of a long lived precursor which regenerates them. Once again there appears to be an exception in the case of Lead-210: if a notable part of the Radon-222 generated by Radium-226 escapes from the dump, Lead-210 will eventually be reduced to less than the amount which would be in equilibrium with radium. There are no substantial clues regarding

*In the Elliot Lake area one producer has been recovering thorium as well as uranium for the last few years, but perhaps not quite on a 100% basis. This will reduce somewhat our estimate of total thorium in waste, but the effect will be ignored for purposes of a low order approximation.

such emanation loss, but if we assume that as much as half the Radium-226 is ineffective in regenerating Lead-210 in the dump, which seems a liberal allowance, then it can be shown that even in 20 years the Lead-210 activity would not have sunk below 85% of the equilibrium value. For the short term we shall therefore conservatively assume that Lead-210 is present at a level corresponding to 75% equilibrium.

Table II incorporates the result of the foregoing assumptions, together with additional information.

TABLE II

ESTIMATED SIGNIFICANT RADIOACTIVITY IN ELLIOT LAKE DUMPS AT THE END OF 1964
(ASSUMING NEGLIGIBLE LOSSES TO HAVE OCCURRED)

Radioisotope			Quantity in Dumps			Chemical Form (guess)	Solubility Gm-Atom/l of activity	Remarks
Name and $\frac{1}{2}$ -Life (yrs)	pc/ μ gm	mpc* _w	Curies	Grams	Gram-Atoms			
Ra-228 6.7	$2.3 \cdot 10^8$	30	1,050	4.5	$2 \cdot 10^{-2}$	RaSO ₄ ?	$5 \cdot 10^{-8}$	Solubility based on book value for RaSO ₄ .
Ra-226 $1.62 \cdot 10^3$	$1.0 \cdot 10^6$	10	12,600	$1.29 \cdot 10^4$	57.	RaSO ₄	"	
Pb-210 19.4	$8.6 \cdot 10^7$	200	9,500	110	0.5	PbSO ₄	$1.0 \cdot 10^{-4}$	Solubility based on book value for PbSO ₄ .
Pa-231 $3.32 \cdot 10^4$	$4.5 \cdot 10^4$	900	585	$1.3 \cdot 10^4$	56.	?	?	
Ac-227 21.6	$7.3 \cdot 10^7$	2000	585	8.0	$3.5 \cdot 10^{-2}$	complex sulphate ?	$5 \cdot 10^{-2}$?	- if similar to La, Gd
Th-232 $1.39 \cdot 10^{10}$	0.11	2000	1,050	$9.5 \cdot 10^9$	$4.1 \cdot 10^7$	partially hydrated sulphate?	0.1?	- solubility based on lab determination of a particular sample
Th-230 $8.0 \cdot 10^4$	$1.94 \cdot 10^4$	2000	12,600	$6.5 \cdot 10^5$	$2.8 \cdot 10^3$	Th(SO ₄) ₂ ·mH ₂ O		Th(SO ₄) ₂ ·3.2H ₂ O.

*1/10th of the "168-hour week, occupational, bone-critical" value of the 1959 ICRP mpc, expressed in picocuries of isotope/litre of water.

Leaching by Contact Waters

Table II provides the base from which to speculate on the activities which may get into the water. The important data are the relative numbers of gram-atoms of each kind of activity available and their specific activities. No quantitative use can be made of the rather meagre solubility figures, since in the first place we are not sure of any of the chemical forms, and in the second the dissolution of slightly soluble substances in a molecule- or ion-trapping milieu cannot be expected to follow laboratory rules. It has been pointed out elsewhere* that radium in ground water fails, by many orders of magnitude, to reach concentrations permitted by the solubility rules, even after supposedly **ample** contact time. Still, the large differences in solubility suggested in the table are of qualitative interest.

A general assumption in the following discussion is that the waste possesses a certain coarse homogeneity, so that contact waters, wherever they lap it, will see the same radioactive inclusions in the same chemical forms.

The Radiums

We see from the mpc column that three picocuries of Ra-228 are worth one picocurie of Radium-226. The mixture rule therefore tells us that in the presence of n picocuries/litre of Radium-228, the mpc for Radium-226 must be reduced from 10 to $(10-n/3)$ pc/l. The question is, will there ever be enough Radium-228 in the water, i.e., will " n " ever be big enough, to matter?

If both radiums were in exactly the same chemical form they would no doubt leach out in the ratio of their gram-atom quantities. This would result in activity concentrations in the ratio of the curie quantities available, that is in the ratio 1,050 to 12,600 or 1 picocurie of Radium-228 to 12 picocuries of Radium-226. This would depress the Radium-226 mpc only 3%, which is negligible. Even if the thorium to uranium ratio in the ore were 1 to 1, as it may well be in the Bancroft area, instead of 1 to 4, the resulting four-fold rise in the curie ratio between Radium-228 and Radium-226 would only depress the Radium-226 mpc by about 12%, which is still unimportant.

Actually a growing proportion of the Radium-228 in the Elliot Lake tails (about 40% at the end of 1964) is the result of regeneration from Thorium-232, and this part may accordingly find itself in a somewhat different chemical form from **that** peculiar to the Radium-228 laid down by the mill. If this part should enter the water more freely (because the thorium complex is rather soluble) the Radium-228 to Radium-226 ratio in water would be increased. If, in this Elliot Lake case for example, it should happen that the tailings-generated Radium-228 goes into solution at a molar rate ten times that of the mill-processed Radium-228, the Radium-226 mpc would be depressed by about 15%, still not very much.

*e.g., A. B. Tanner, Chapter 14 of "The Natural Radiation Environment", University of Chicago Press.

The proportion of regenerated Radium-228 in the dumps increases with time, as stated above. If mine production continues constant, the mill-processed Radium-228 approaches a finite upper limit (about 1000 curies in Elliot Lake dumps if the average production rate of the last ten years is maintained) while the regenerated Radium-228 increases in step with the deposition of Thorium-232. If production ceases, the mill-processed Radium-228 dies away in the dumps, while the regenerated Radium-228 maintains a constant level indefinitely into the future.

We might therefore venture as a general conclusion that Radium-228 is probably not a serious toxic competitor to Radium-226 in the affected Elliot Lake area waters at the present time. If however there is any ground for suspicion that the higher solubility of the thorium complex carries over to contained Radium-228, this can only augment the relative importance, present and future, of the Radium-228. The trend of the argument suggests that an exploratory investigation would be justified, perhaps with emphasis on the affected waters of the Bancroft area, where the thorium runs higher.

Occurrence can perhaps be expected to be spotty, because of the very low mole quantities available.

Lead-210

The table tells us that 20 picocuries of Lead-210 are worth one picocurie of Radium-226. Since the former is a descendant of the latter, the respective mole quantities in the dump should stay roughly in step, subject to variations in the loss rate of the radon intermediate. If the effective molar dissolution rates of lead sulphate and radium sulphate were the same we should expect to see a Lead-210 to Radium-226 ratio in the water of about 9,500 to 12,600 or about 1 picocurie to 1.3 picocuries. This would depress the Radium-226 mpc only by about 3%. However, the laboratory solubility of lead sulphate is noticeably higher than that of radium sulphate, so the Lead-210 may well be enhanced in solution. An enhancement by a factor of only 5 or 6 would begin to be interesting. Only subsequent analyses will tell the tale.

Protactinium-231 and Actinium-227

The chemical facts of life relating to protactinium are not sufficiently known to permit any guesses, even of the kind we have been making here. Protactinium-231 has been left in the table to show that substantial mole quantities are present in the dumps, so that if an effective leaching mechanism existed it could be troublesome. We should begin to be interested at levels of a few hundred picocuries, or of about 0.01 microgram, per litre. If Actinium-227 is present as a fairly normal sulphate, its solubility may give it some interest. At the moment we are not prepared for analytical work on either of these.

The Thoriums

It takes 200 picocuries of either Thorium-232 or Thorium-230 to be worth one picocurie of Radium-226. However, there are ample mole quantities of both in the dumps, and the chemical complex in which they occur has been established (by analysis of slime supernatants, etc.) as rather soluble. It is therefore not impossible that we may find the relatively lower toxicity of the thoriums counter-balanced by a relatively higher concentration in water.

If the mill treatment extracts Thorium-230 and Thorium-232 with equal efficiency from their respective mineral complexes they should come out in the same chemical form. Both are long-lived and neither is regenerated in the dump. As long as the thorium to uranium ratio stays the same in the ore, the gram-atom and curie ratios of the two thoriums in the dump will remain constant. This has the result that one microgram of composite thorium element from the Elliot Lake dumps carries 0.11 picocurie of Thorium-232 activity and $12 \times 0.11 = 1.3$ picocuries of Thorium-230 activity or a total of 1.4 picocuries. Thus a thorium concentration of $2000/1.4 = 1400$ micrograms per litre constitutes an mpc in the absence of radium or other competitors. This is six or seven times lower than the microgram per litre mpc value for so-called natural thorium. In view of the solubility of the thorium complex, concentrations of a few hundred micrograms per litre would not be unexpected in some of the affected waters. The Radium-226 mpc would be depressed 10% for every 140 micrograms of thorium present in a litre of water.

In the Bancroft area on the other hand the likely ratio of the two thoriums favours a larger mpc in terms of micrograms per litre, although what this is has not been worked out.

Summary and Conclusions

It has been pointed out that radiological pollution of waters from mill tailings in the uranium mining areas of Ontario has, in the past, been depicted almost exclusively in terms of Radium-226 concentrations. While this radioisotope is undoubtedly the most important, we should not set the final objectives for radiological water quality in complete disregard of the marginal effect of others. Six of these others have been picked out as being worth further study, and some general arguments have been advanced in support. Thorium may be the most important extra in the Elliot Lake area waters, and it is particularly noted that this is not "natural thorium" as defined by the ICRP but a composite having, in gravimetric terms, a considerably lower mpc. We have used for purposes of argument the 1959 ICRP population mpc values without prejudice to any future decision to adopt these or still later ones. On the 1959 basis, the mpc for Radium-226 in the affected Elliot Lake area waters would be reduced 10% by the presence of any of the following: 3 pc/l of Radium-228; 20 pc/l of Lead-210; 90 pc/l (or 0.002 $\mu\text{gm/l}$) of Pa-231; 200 pc/l of Ac-227; and by 140 $\mu\text{gm/l}$ of thorium."

In relation to the contamination of water by radioactive wastes, particularly Radium-226, it is of interest to note that most natural waters have a Radium-226 content of less than 1 picocurie per litre (pCi/litre) and many contain less than 0.1 picocuries per litre.

The following statements are conclusions from the 1965 Deputy Minister's Report entitled "Report on Radiological Water Pollution in the Elliot Lake and Bancroft Areas - 1965":

- (1) In accordance with modern scientific knowledge and opinion, the Department considers that any unnecessary exposure to radioactivity should be kept to a minimum.
- (2) With respect to water quality standards for radioactivity, there is no single figure value which distinguishes between safe and harmful levels of radioactive content.
- (3) The Department recommends that levels from 30 to 10 picocuries of Radium-226 per litre of water be adopted as the immediate or first-level objective to be attained in those lakes and streams where present levels are in excess of this range.
- (4) The Department recommends that levels from 10 to 3 picocuries of Radium-226 per litre be adopted as the immediate objective to be attained in public drinking waters in the Elliot Lake and Bancroft areas.
- (5) The Department does not consider that such exposure to Radium-226, as would be entailed in swimming or part-time use of lakes containing

between 20 and 40 picocuries per litre, presents any hazard to health.

Since 1964, the Ontario Water Resources Commission has set the objective for Radium-226 in drinking water supplies at 3 picocuries per litre, and we are also attempting to reduce industrial effluents to this objective.

CHRONOLOGICAL HISTORY OF WASTE TREATMENT AT STANROCK URANIUM MINES LIMITED

1957

The Stanrock mine had not come into production. Can-Met (in partial production) was the sole contributor to the joint Stanrock-Can-Met tailings area. In theory, Can-Met was adding lime to the effluent in order to control pH. The chemical characteristics of the tailings area effluent during the 1957 period were as follows:

Total Solids	426 ppm
Susp. Solids	26 ppm
Diss. Solids	400 ppm
pH	4.5
Iron as Fe	0.2 ppm
Sulphate as SO_4	248 ppm
Nitrogen (Nitrate)	0.30 ppm

1958

During 1958, Stanrock Uranium Mines Limited came into production. Tails and the acidic ion exchange barren were treated in leach agitators by an automatically-controlled addition of lime. After "neutralization", these wastes were combined and pumped as tailings to the area set aside to receive them.

Can-Met continued to discharge to the same tailings area.

The chemical characteristics of the tailings area decant during the 1958

period were as follows:

Total Solids	6,188 ppm
Susp. Solids	2,326 ppm
Diss. Solids	3,862 ppm
pH	4.4
Iron as Fe	200 ppm
Sulphate as SO_4	1,943 ppm
Nitrogen (Nitrate)	28 ppm

1959

Operations and waste treatment continued as in 1958. The barren solution was being treated in a set of three neutralization tanks that were equipped with agitation mechanisms. A separate but similar set of three tanks was used to treat the tailings slurry. Automatic pH recorders, used to regulate the amount of slurried lime added, were set at approximately 7 for both batteries of tanks. After neutralization, the barren solution and tailings slurry were combined and, as a unit, discharged to the tailings area.

The chemical characteristics of the tailings area decant during the 1959 period were as follows:

Total Solids	4,010 ppm
Susp. Solids	98 ppm
Diss. Solids	3,912 ppm
pH	3.0
Iron as Fe	84 ppm
Sulphate as SO_4	unknown
Nitrogen (Nitrate)	26 ppm

Can-Met was still in operation in 1959.

1960 - 1964

In April 1960, Can-Met suspended milling operations. Operations at Can-

Met to date (1970) have not been resumed. Stanrock continued normal operations with only minor changes in the waste treatment system. In October 1964, Stanrock Uranium Mines Limited ceased conventional underground mining and, as such, solid tails ceased to be discharged to the tailings area. Bacterial leaching became the prominent mining procedure at the property. Neutralized barren solution continued to discharge to the tailings area.

1965

In 1965, it was reported that the total flow of barren solution from ion exchange totalled 120,000 gallons per day. Of this total, 30,000 gallons were being returned underground while the remaining 90,000 gallons were being neutralized and then discharged to the tailings area.

In May of 1965, in order to provide a measure of radium control in the tailings area, the addition of barium chloride to the barren solution was commenced. Initially, .005 pounds of $BaCl_2$ /ton of barren was added. A final pH of 8.0 was achieved by adding lime to the treated barren solution.

1966

$BaCl_2$ addition and neutralization of the barren solution continued.

1967 - 1970

In 1967, the barren solution being discharged to the tailings area was re-directed underground. Since that time, industrial waste discharges (if any) to the tailings area have been rare.

During this period of time, seepage from the tailings area became a major problem. With the exception of the North Dam Seepage, treatment has not

been provided for the wastes from the property since 1967. BaCl_2 addition was discontinued when the barren solution was entirely re-directed underground.

The above brief waste treatment summary tends to give the impression that, at least during the years 1960 - 1966, the pH of the wastes leaving the Stanrock operation met the Objective of the Ontario Water Resources Commission for discharge to a natural watercourse. This is strictly not true. This fact is clearly indicated by the following summary of pH determinations performed on samples of the tailings area decant over the years.

Tailings Area Decant (pH readings)

<u>Year</u>	<u>Month</u>	<u>Day</u>	pH (at Lab.)	Deviation from OWRC Lower Objective (pH = 5.5)
1957	October	24	4.6	-.9
1958	December	2	4.4	-1.1
1959	June	25	3.0	-2.5
	June	28	3.2	-2.3
1960	June	23	4.2	-1.3
1961	July	13	3.7	-1.8
	July	19	4.0	-1.5
	July	21	3.9	-1.6
	August	4	5.1	-0.4
	October	4	5.2	-0.3
	November	1	5.6	+0.1
	November	10	4.0	-1.5
	December	14	4.4	-1.1
1962	January	11	4.3	-1.2
	February	19	5.4	-0.1
	March	15	5.6	+0.1
	April	13	6.5	+1.0
	May	14	6.1	+0.6

<u>Year</u>	<u>Month</u>	<u>Day</u>	pH (at Lab.)	Deviation from OWRC Lower Objective (pH = 5.5)
1962	June	21	5.3	-0.2
	July	16	6.4	+0.9
	August	10	4.3	-1.2
	September	18	4.8	-0.7
	October	12	6.2	+0.7
	November	16	3.7	-1.8
	December	17	3.9	-1.6
1963	January	14	6.9	+1.4
	February	12	3.2	-2.3
	March	12	6.7	+1.2
	April	11	6.8	+1.3
	May	7	3.1	-2.4
	June	12	3.2	-2.3
	July	3	3.2	-2.3
	August	14	2.8	-2.7
	September	11	8.4	+2.9
	December	11	5.2	-0.3
1964	January	7	8.4	+2.9
	February	12	4.1	-1.4
	April	--	8.3	+2.8
	May	15	4.0	-1.5
	June	15	2.9	-2.6
	September	9	5.1	-0.4
	October	--	5.9	+0.4
	November	11	7.8	+2.3
	December	--	8.2	+2.7
1965	January	--	8.0	+2.5
1966	April	20	4.3	-1.2
	August	17	3.0	-2.5
1967	January	5	9.0	+3.5
	May	6	6.9	+1.4
	July	12	3.4	-2.1
	August	30	3.4	-2.1

<u>Year</u>	<u>Month</u>	<u>Day</u>	pH (at Lab.)	Deviation from OWRC Lower Objective (pH = 5.5)
1968	August	12	2.7	-2.8
	September	20	2.8	-2.7 (heavy rain)
1969	June	18	2.0	-3.5
	October	21	2.8	-2.7

Of the above 55 samples taken over the past thirteen years, 37 indicate that the waste was not acceptable with regard to pH for discharge to a natural watercourse.

For the past two years, the pH of the tailings area decant has, without exception, fallen within the range pH = 2.0 - 3.0.

Lime additions, even when the mine-mill was in full operation, have rarely been adequate to maintain minimum water quality standards in the area.

A reflection on the efficiency of the tailings area as an impoundment basin can be gained very simply by reviewing the suspended solids concentrations found in the tailings area decant over the past thirteen years.

NOTE: The Commission's Objective (for discharge to a natural watercourse) with regard to suspended solids is 15 parts per million.

Tailings Area Decant (conc. Susp. Solids)

<u>Year</u>	<u>Month</u>	<u>Day</u>	<u>Conc. Susp. Solids (ppm)</u>
1957	October	24	26
1958	December	2	2,326
1959	June	25	98
	June	28	468

<u>Year</u>	<u>Month</u>	<u>Day</u>	<u>Conc. Susp. Solids (ppm)</u>
1960	July	29	76
	August	8	290
	December	27	442
1961	January	23	378
	February	10	570
	March	10	340
	March	25	3,062
	April	3	998
	April	10	468
	June	7	274
	June	22	276
	June	23	440
	July	19	228
	November	1	666
	December	14	1,200
1962	January	11	236
	February	19	904
	March	15	2,330
	April	13	356
	May	14	352
	June	21	13
	July	16	2
	August	10	130
	October	12	22,608
	November	16	122
	December	17	81
1963	January	14	214
	February	12	345
	March	12	88
	April	11	19,188
	May	7	131
	June	12	21
	July	3	134
	August	14	148
	September	11	2
	December	11	7
1964	January	7	1
	February	12	82

<u>Year</u>	<u>Month</u>	<u>Day</u>	<u>Conc. Susp. Solids (ppm)</u>
1964	June	15	82
	September	9	30
	October	4	3
	November	11	156
1965	January	--	32
1966	April	20	15
1967	January	5	30
	May	6	9
	July	12	136
1968	August	12	95
	September	20	224 (heavy rain)
1969	June	18	25
	October	21	170

To our knowledge, special chemical flocculants (settling aids) were never used to improve sedimentation in the Stanrock tailings area. If such aids were used, they were definitely not added in sufficient quantities.

THE BEGINNING OF POSITIVE WASTE TREATMENT AT STANROCK URANIUM MINES LIMITED

In June of 1968, continuous neutralization of the North Dam Seepage (see section entitled "Sources of Liquid Wastes" - Section C, Tailings Area - Subsection F, North Dam Seepage) commenced. At first, a few loads of discard lime were dumped into the ditch carrying the seepage. The effect of this action on the waste was negligible. The mine personnel soon discovered that lime had to be in a finely divided state in order to achieve the required results. Shovelfulls of discard lime thrown into the ditch reacted immediately on contact with the

waste. The most obvious reaction product was dense olive-green masses of ferrous hydroxide.

An advanced stage of waste treatment was provided when a lime line running from the mill to a rocky ledge overlooking the minor spill area (created on April 17, 1964 when a section of one of the tailings dams collapsed) was installed. The lime slurry, pumped continuously from the mill, was initially too weak to accomplish more than a minor amount of neutralization. The precipitation of ferrous iron and its subsequent oxidation were thus being accomplished only on a limited scale.

Photograph V

The initial attempt to continuously neutralize the North Dam Seepage. The minor spill area is in the background. The photograph clearly shows the precipitation of ferrous iron on contact with the lime solution.

Subsequently, the lime slurry discharge was increased in flow and at the same time the line was moved closer to the mill and, in fact, to the very "headwaters" of the North Dam Seepage. The lime slurry (with a pH of 12.8 and a flow of approximately 25 - 50 gallons per minute) began to influence the North Dam Seepage greatly. The pH of the waste came under control and as a result the ferrous iron (which generally makes up over 50 per cent of the total iron content of the waste) began to precipitate in vast quantities. The subsequent oxidation to ferric iron was also observed to be accelerated to an impressive extent.

The uninformed viewer would perhaps shudder to see the effects of neutralization.

Upstream of the lime discharge, the North Dam Seepage is generally clear and sparkling in appearance. Its dissolved solids content of nearly 20,000 parts per million is not apparent. Its iron content of up to 3,300 parts per million rarely colours the waste at all. And yet, this innocuous seepage stream and the many others like it in the Elliot Lake district constitute an extremely toxic waste in which only a few species of bacteria can survive.

The character of the seepage stream is revealed immediately upon lime addition. The waste stream becomes dark green and opaque in appearance.

Photograph W

A portion of the North Dam Seepage after neutralization.

Slow moving portions of the stream become so choked with the malachite-coloured precipitate that the effluent, which on occasion acts as a semi-gelatinous substance, can be picked up in the hands.

Varying distances downstream, and occasionally before the waste reaches Quirke Lake, the dense green precipitate turns rusty-orange in colour indicating that at least some of the iron has completed its oxidation to the ferric state. If the waste has to be discharged to a watercourse, it is technically preferable that the waste be in this "oxidized" state.

Unfortunately, in 1969, the North Dam Seepage was the only waste stream being neutralized. After neutralization, the waste was joined by the Swallow Hill Seepage and the overflow from Beaver Lake creating, in effect, a deficiency of lime so that large quantities of ferrous iron were still gaining access to Quirke Lake.

With little retention time available in the watercourse, most of the iron precipitate (ferrous iron and ferric iron) was being swept into the lake.

Photograph X

The neutralized North Dam Seepage after union with seepage from Swallow Hill.

A further important step with regard to waste treatment at Stanrock was

taken when mine management decided to construct a crib dam on the downstream side of the minor spill area. The purpose of this dam is to create a fairly large impoundment area for the North Dam Seepage. Fortunately, the location of the dam is such that the entire waste flow from the Swallow Hill area will also be collected in the impoundment area. A pump is to be installed on the top of the dam and will serve to transfer the impounded wastes from the dam site to the main tailings area. Lime addition (to the North Dam Seepage) will be continued as long as possible and therefore the impoundment area will serve to trap and hold the various precipitates of the metals involved. Some of the metals will not, of course, form precipitates under the conditions that are to be set up.

The construction of the crib dam began in the fall of 1969. Completion is expected by the spring of 1970.

Photograph A1

The crib dam under construction in 1969. The Mine Manager, Mr. B. G. MacDermid, is seen standing on the unfinished upstream side of the dam.

It is expected that the crib dam will put the entire minor spill area under water.

It appears that economic conditions will force the suspension of the lime treatment in the near future but, with the continuous pumping of the waste to the tailings area (i.e., the waste will not be directed to Quirke Lake), this condition can be tolerated provided that the schedule for waste treatment at Stanrock (explained in full in a later section of this report) is closely followed.

With the completion of the new crib dam and the subsequent process of waste pumping to the tailings area, the only effluent (ignoring the compressor cooling water) to Quirke Lake from the entire Stanrock property will be the highly contaminated overflow from Beaver Lake.

As recommended by the field staff of the Ontario Water Resources Commission, mine management have agreed to attempt the diversion of Beaver Lake (in the spring of 1970) from Quirke Lake to the Lower Serpent River watershed via Orient Creek. This diversion can be accomplished fairly easily and, when completed, will mean that wastes from Stanrock Uranium Mines Limited will no longer gain access to Quirke Lake.

Mine management to date (1970) have not indicated any further plans for waste treatment on the Stanrock property.

COMMENTS CONCERNING THE DIRECTION OF FLOW OF WASTES FROM THE VICINITY OF STANROCK URANIUM MINES LIMITED AND THE DECISIONS INVOLVED

As stated earlier in this report, the Stanrock tailings area is the largest and most complex single source of pollution in the Elliot Lake district. Because of the geographical position of the tailings body, highly contaminated wastes have been continuously gaining access, not only to Quirke Lake (a part of the Upper Serpent River System), but also to the Lower Serpent River system. The complex drainage system of the area and the rugged terrain makes it almost economically impossible to completely or even partially treat, on an individual basis, the many waste streams involved.

(i) The First Decision

Stanrock Uranium Mines Limited should be required to combine all waste flows that have resulted directly or indirectly from past and/or present mining operations in such a manner that a single waste stream flowing in one direction and to only one watershed results.

Having made the above decision, it becomes necessary to determine which route the wastes would take in order to reach Pecors Lake and hence, the Lower Serpent River system (see Diagram P-58; also see Sheet #3 in back pocket).

Two routes were available:

Route #1

The primary waste flow would be directed to Quirke Lake. Quirke Lake overflows to three large lakes (Nook, Kindle and Whiskey) in series. Whiskey Lake overflows to Pecors Lake (the "common" lake of both routes).

Quirke Lake, with a surface area of 4,600 acres and a maximum depth of 100 meters is one of the largest and most beautiful lakes of the Serpent River watershed. The quality of its water has been seriously impaired by the surrounding mining operations (see Sheet #4 in back pocket) but not to the extent that the lake cannot recover naturally. Water from Quirke Lake is used by some of the operations as industrial make-up water. Curiously enough, some of the mines using the water have already complained to the Ontario Water Resources Commission about the poor quality of the water even though they are primarily responsible for it.

Quirke Lake is readily accessible to the public and its proximity to the Mississagi Provincial Park makes it a very desirable attraction in the area.

Nook, Kindle and Whiskey Lakes are still in fairly good condition although this is due solely to their relative remoteness from the mining operations in the vicinity of Quirke Lake.

It is therefore desirable that the Quirke-Nook-Kindle-Whiskey route not be disturbed and that every effort be made to control the discharges into the lake. (Quirke)

Route #2

The primary waste flow would be directed to the Lower Serpent River system via Pink Dragon Lake, Half Moon Lake, May Lake and Hough Lake. Hough Lake overflows to Pecors Lake (the "common" lake for both routes).

As indicated in a previous section of this report, Pink Dragon Lake and Half Moon Lake have been acting solely as waste treatment facilities for the Stanrock operation. As far as lake ecology is concerned, both Pink Dragon Lake and Half Moon Lake should be regarded as absolute disaster areas.

May Lake and Hough Lake are both subject to acidic runoff from another operation as well as from Stanrock and, as such, both are very seriously impaired. The Commission therefore is of the opinion that an increased waste discharge to the Quirke Lake route would cause a more noticeable and serious change in water quality than would an equivalent increase in waste discharge to the Pink Dragon

route.

(ii) The Second Decision

Stanrock Uranium Mines Limited should be required to direct all existing waste flows on the property and in the vicinity of the property that have resulted from past and/or present mining operations to the Lower Serpent River watershed via Pink Dragon, Half Moon, May and Hough Lakes. Under no circumstances should waste flows (seepage or otherwise) be directed to Quirke Lake or to the Quirke Lake watershed.

COMMENTS - THE IMPLEMENTATION OF THE FIRST AND SECOND DECISIONS

(a) The new crib dam structure has been equipped with decant mechanisms. These decant mechanisms must be kept sealed at all times. An overflow or any continuous seepage from the new crib dam to Quirke Lake must be prevented.

Impounded waste is to be pumped back onto the main tailings area. This scheme is entirely acceptable but may put extra stress on the decant structure in the main tailings area. Since it is the intention of the mine to discharge this waste into the vicinity of the decant pond using sections of an old wood-stave tailings line, most of the waste will flow over the decant rather than being re-adsorbed by the tailings mass.

It is not required but is preferable that the impounded North Dam Seepage, etc., be pumped into Beaver Lake or to an area where the waste could run by gravity into Beaver Lake.

(b) The diversion of Beaver Lake into the Orient Creek Swamp, and hence, into

Orient Creek and Pink Dragon Lake must be completed. The diversion can be easily accomplished by digging a trench from Beaver Lake through the small ridge that currently separates Beaver Lake and the Orient Creek Swamp.

NOTE: The ridge in question is three or four feet high at its highest point and only a few feet wide. The length of the ditch will probably not exceed two hundred feet. The ground in the area is generally swampy earth. In the immediate vicinity of the ridge, bedrock may be encountered at a depth of two or three feet. In this case, blasting powder will have to be used.

If possible, the ditch should be cut at least a foot below the base level of the beaver dam that now controls the upper foot or so of water in Beaver Lake. If the water level does not drop below the level of the base of the beaver dam, the dam should be completely sealed with earth. If the diversion ditch cannot be economically driven through the bedrock encountered, then the beaver dam at the other end of the lake should be built up and completely sealed with earth.

Wastes from the beaver dam area must not be allowed to reach Quirke Lake.

NOTE: The field staff of the Ontario Water Resources Commission have inspected carefully the entire route (from Beaver Lake to the major spill area) of the proposed diversion. The waste as a result of the diversion will not gain access to Quirke Lake but will flow into Orient Creek and hence, into Pink Dragon Lake.

In order to further verify the direction of flow along the new route, small sections of two beaver dams located in the Orient Creek Swamp were destroyed. The ensuing rush of water, after an interval of approximately half an hour, arrived at the mine road following the course of Orient Creek, flowed through the culvert

under the mine road, and discharged onto the major spill area. The flow was sufficient to almost wash across the mine road. On this basis, we recommend that the right-angle bend Orient Creek makes on meeting the mine road be eliminated and replaced by a gentle curve (this operation should take less than an hour).

ADDITIONAL WASTE TREATMENT FACILITIES REQUIRED

The continuing deterioration of the decant facilities that serve the South Dam, the deterioration of the tailings area itself, and the probable rapid mass oxidation of the tailings in the major spill area all indicate that a low dam should be constructed across the downstream end of the major spill area as rapidly as possible. The dam will serve the following purposes:

- (a) to flood the entire major spill area to a depth of at least one foot in order to slow down the mass oxidation of the tailings in the area;
- (b) to collect the entire flow from the Stanrock operation so that it can be monitored and treated as a unit;
- (c) to retain the tailings that are continuously being washed down Cinnabar Creek into the major spill area, and to prevent tailings from the major spill area from washing into Pink Dragon Lake;
- (d) to provide a downstream safety factor with regard to the stability of the South Dam, etc.;
- (e) to prevent toxic surges of waste (due to heavy rainfall over the tailings area, etc.) from reaching Pink Dragon Lake and Half Moon Lake; and
- (f) to provide sufficient retention time so that the natural oxidation of ferrous

iron to the ferric state can be at least initiated (as is now the case in Beaver Lake, Pink Dragon Lake, and the Decant Swamp). Even without lime addition, a small quantity of the ferrous iron will be oxidized to the ferric state and thus cause a slight relief in downstream loadings.

The dam, as such, will not have to be very high - nor will it have to be impermeable. In order to allow the wastes to escape, it must be equipped with an overflow mechanism such as a wooden sluiceway or a gate of logs, etc.

The downstream end of the major spill area is fairly narrow and, as such, the actual length of the dam will not be excessive. The materials of construction could quite easily be the dry tails in the area itself along with the black peat that underlies the area. If possible, the dam should be capped with the black mud in the area and seeded.

The area of construction is easily accessible by bulldozer. The field staff suggests that any heavy equipment that is to be taken into the area follow a route along the edge of the tailings mass rather than attempt to cut across it. Open water (up to four feet deep) underlies some isolated sections of the interior of the major spill area. In these instances, one or two foot thick layers of tailings are held up by a dense mat of floating organic material.

In order to achieve complete waste control at the property, lime or its equivalent will have to be added continuously to the waste ponding behind the required new dam.

Radiological control could be achieved at the same time by the addition

of BaCl_2 to the influent to the area.

Neutralization facilities, etc., could probably be set up to advantage on Cinnabar Creek at the mine road. The resulting precipitates of the metals would settle conveniently in the spill area to form a stable coating over the tailings mass.

GENERAL CONCLUSIONS

- (1) Stanrock Uranium Mines Limited is solely responsible for the present serious impairment of Beaver Lake, Pink Dragon Lake, and Half Moon Lake. The Company is partially responsible for the impairment of Quirke Lake, May Lake, Hough Lake, and Pecors Lake.
- (2) The value of Stanrock production during the period 1958 - 1967 has been given as 115 million dollars. Even during its most profitable years, this Company managed to display a complete disregard for the maintenance of accepted water quality standards in the region of its operation. At present, only a minimum effort is being made by the Company to control its waste discharges.
- (3) At present, Stanrock Uranium Mines Limited appears to be engaged in an attempt to "wait out" the current slump in the uranium-rare earths market. This "waiting" is being done at the expense of the general public.

RECOMMENDATIONS

- (1) The crib dam to contain the North Dam and Swallow Hill seepage flows should be completed immediately along with the associated pumping facilities. The waste should be directed back to the main tailings area or preferably to

Beaver Lake. There should be no flow over the dam or through it.

- (2) Beaver Lake should be completely diverted into Pink Dragon Lake and hence, into the Lower Serpent River watershed. There should be no overflow from Beaver Lake to Quirke Lake.
- (3) A dam (as described in the section of this report entitled "Additional Waste Treatment Facilities Required") should be built across the downstream end of the major spill area.
- (4) Complete treatment (chemical and radiological) of the wastes ponding behind the dam to be constructed at the downstream end of the major spill area should be provided.

Prepared by:

...*J. R. Hawley*.....
J. R. Hawley,
Mining Technician,
Senior Metallurgical Technician,
Chemical Technologist,
Division of Industrial Wastes.

Approved by:

...*R. C. Stewart*.....
R. C. Stewart, P. Eng.,
District Engineer,
Division of Industrial Wastes.

JRH/oc

A P P E N D I X A

CAN-MET EXPLORATIONS LIMITED

The Can-Met mine is on the south shore of Quirke Lake about 16 miles by road from the Town of Elliot Lake.

Two shafts were sunk on the property. The first was a service shaft that bottomed at 2,126 feet while the second, a three-compartment shaft, terminated at a depth of 2,395 feet. Both shafts were connected by a haulageway at the 2,100 foot horizon.

A three thousand ton daily capacity mill was constructed on the property. Plant construction was begun on July 31, 1956, and completed on August 15, 1957. Owing to miscellaneous delays, production did not begin until October 21, 1957. The initial rate was 900 tons of ore per day. The first shipment of uranium precipitates was made in December (1957). Delays were encountered in shaft-sinking owing to water conditions and the scarcity of highly skilled shaft-sinking crews. Later, underground development was held up due to a badly faulted orebody.

Ore reserves were reported by the Company in 1959 to be 6,642,380 tons of partly proven ore with an average grade of 0.092% U_3O_8 (10% dilution allowed for). The total tonnage, including all categories of ore, was reported to be 8,362,069 tons (Company prospectus, April 2, 1958).

The Company received a contract from Eldorado for the sale of precipitates valued at \$75,852,000. This was later increased to \$79,670,000, and the delivery time was extended from March 31, 1962 to March 31, 1963.

NOTE: In October 1959, Can-Met announced that underground rock conditions made

cost-cutting procedures less effective and that a low treatment rate resulted in higher unit costs. Soon thereafter it was announced that ore reserves were insufficient to fulfill contract commitments, and on March 24, 1960, the Company amalgamated with Consolidated Denison Mines Limited to form a new company - Denison Mines Limited.

Mining operations at Can-Met ceased in April 1960 (milling ceased April 15, 1960), and the remainder of the Can-Met contract was transferred to Denison.

The mill was sold in June 1962.

Geology

The Can-Met deposit is on the north limb of the Quirke Lake syncline. It is bordered on the west by Denison, on the north by Panel and on the south by Stanrock. The depth of the ore (which does not outcrop on the property) is from 1,500 feet in the north to 2,000 feet near the shafts. The average strike is north 40 degrees west, with dips ranging between 0 and 15 degrees south. The average thickness of the ore is 17 feet, but widths of up to 25 feet have been encountered. Extensive faulting occurs throughout the orebody.

The orebody, which consists of two beds of uraniferous quartz-pebble conglomerate, is in one of the fringe areas of the main ore channel situated on the north side of the Quirke Lake Trough. The beds rest on the basement rocks at the north end and 70 feet above the basement at the south end.

A sample of ore sent to the Mines Branch in 1956 was found to contain brannerite, monazite and uraninite. Possible by-products were reported by a Company official to be thorium and rare-earth minerals. The ratio of uranium to

thorium was said to be 2:1. Large pockets of pyrrhotite, pyrite and some chalcopyrite have been encountered in the ore bed.

NOTE: The ore was recovered utilizing the conventional room-and-pillar method of mining. As such, 30% of the ore remains in place and can probably be recovered at a future date. The pillars are about 20 feet square.

Milling

The treatment plant went into operation on October 21, 1957, at an initial rate of 900 tons per day. The first shipment of yellowcake was made on December 9, 1957 and in September 1958, the mill reached its capacity of 3,000 tons of ore per day.

The plant was of the conventional acid-leach, ion-exchange, precipitation type. During the early part of 1959, recovery was 93%; mill heads were 0.103% U_3O_8 . The precipitate was shipped directly to the United States.

All tailings were discharged to the tailings basin on Stanrock property.

Production

The following, taken from the Company's annual report for 1958, is a summary of production for the period October 16, 1957 to September 30, 1958:

Tons ore broken	- 541,296
Tons ore hoisted	- 531,898
Dry tons milled	- 535,466
Average daily tonnage	1,530

% extracted	-	91.71
% recovered	-	90.83
Pounds U_3O_8 produced	-	826,250
Average mill head (% U_3O_8)	-	0.092

To September 30, 1959, the mill handled 1,477,160 tons of ore and recovered 2,495,709 pounds of U_3O_8 . Revenue from May 1, 1958 to September 30, 1959 totalled \$23,517,448. Production figures covering the period October 1, 1959 to April 1960 are not available. Can-Met did, however, report that, up to the end of 1959, a few months prior to closure, it had produced 4,719,000 pounds of U_3O_8 out of its contract total of 7,710,600 pounds.

A P P E N D I X B

WATER QUALITY OBJECTIVES

POLICY GUIDELINES

- (1) The provincial water resources will be utilized wisely in the best interests of the people of Ontario. This will require the restoration and maintenance of water quality for the greatest possible use. Towards this end, water quality objectives will take into consideration the use and value of water resources for public, agricultural and industrial water supplies, propagation of fish and wildlife, recreational purposes, aesthetic enjoyment and other legitimate uses.
- (2) There must be a constant effort to improve the quality of water, for it is recognized that the improvement of the quality of water makes it available for more uses.
- (3) Minimum quality control objectives will be set to apply to all waters of the province; more stringent objectives may be set for any individual situation dependent upon use, and in the future, generally more stringent objectives may be set for specific drainage basins or drainage areas.
- (4) In general, beneficial uses of water will be the controlling factors in determining the water quality objectives in any drainage basin. Where the use of water for the assimilation of treated wastes in a properly controlled fashion within a drainage basin is recognized as a reasonable use, it must be compatible with the other uses of that water.
- (5) Economic, health, aesthetic and conservation values which contribute to the social and economic welfare of an area will be taken into account in

determining the most appropriate use or uses of a water resource. Therefore, in the establishment of water quality objectives for specific drainage basins, the opinions of agencies or persons having an interest and/or responsibility in the present or future utilization of the water in a particular basin will be solicited and evaluated.

- (6) For each beneficial use, there are certain water quality requirements which must be met to assure that the water will be suitable for that beneficial use. The co-operative assistance of technically qualified persons who are specialists with regard to various water uses will be sought in determining the requirements.
- (7) Caution should be exercised in selecting numerical values for parameters to be included in any objectives for water quality; only those values will be included for which sound information on applicable levels is available. In the absence of appropriate numerical values, the objectives will consist of verbal descriptions in sufficient detail as to show clearly the quality of water intended.
- (8) Water quality objectives will be revised periodically as new information and conditions develop. Objectives will not be considered final or absolute as increases in scientific knowledge of the effects of wastes on the environment will inevitably require their improvement.
- (9) All wastes, prior to discharge to any receiving watercourse should receive the best practicable treatment or control. Such treatment must be adequate

to protect and upgrade water quality in the face of population and industrial growth, urbanization and technological change.

- (10) Water quality objectives will provide an engineering base for design of treatment works by municipalities and industries. Such objectives will enable municipalities and industries to develop realistic plans for new plants or expanded facilities without uncertainties about waste disposal requirements.
- (11) Effluent requirements based on the applicable water quality objectives for a drainage basin will be established for each user by the Commission, in order to maintain acceptable water quality for all beneficial uses within the drainage basin. Requirements may be revised when necessary to allow for increasing or new uses of the waters of a drainage basin.
- (12) Any user who discharges wastewater that does not meet requirements established by the Commission, or otherwise impairs the quality of the water, will be subject to the provisions of the OWRC Act.

OBJECTIVES FOR WATER QUALITY CONTROL IN THE PROVINCE OF ONTARIO

Adopted by the Ontario Water Resources Commission

These objectives are for all waters in the Province of Ontario, and it is anticipated that in certain specific instances, influenced by local conditions, more stringent requirements may be found necessary.

GENERAL OBJECTIVES:

All wastes, including sanitary sewage, storm water, and industrial effluents, shall be in such condition when discharged into any receiving waters that they will not create conditions which will adversely affect the use of these waters for the following purposes: source of domestic water supply, navigation, fish and wildlife, bathing, recreation, agriculture and other riparian activities. In general, adverse conditions are caused by:

- (a) Excessive bacterial, physical or chemical contamination.
- (b) Unnatural deposits in the stream, interfering with navigation, fish and wildlife, bathing, recreation or destruction of aesthetic values.
- (c) Toxic substances and materials imparting objectionable tastes and odours to waters used for domestic or industrial purposes.
- (d) Floating materials, including oils, grease, garbage, sewage solids, or other refuse.
- (e) Discharges causing abnormal temperature, colour or other changes.

SPECIFIC OBJECTIVES:

In more specific terms, adequate controls of pollution will necessitate

the following objectives for:

(a) Sanitary Sewage, Storm Water, and Waste from Water Craft:

Sufficient treatment for adequate removal or reduction of solids, bacteria and chemical constituents which may interfere unreasonably with the use of these waters for the purpose aforementioned.

Adequate protection for these waters, except in certain specific instances influenced by local conditions, should be provided if the coliform MPN median value does not exceed 2,400 per 100 ml at any point in the waters following initial dilution, and the five-day BOD does not exceed 4 ppm.

(b) Industrial Wastes:

(1) Chemical Wastes - Phenolic Type

Industrial waste effluents from phenolic hydro-carbon and other chemical plants will cause objectionable tastes or odours in drinking or industrial water supplies and may taint the flesh of fish.

Adequate protection should be provided for these waters if the concentration of phenol or phenolic equivalents does not exceed an average of 2 ppb and a maximum of 5 ppb at any point in these waters following initial dilution. This quality in the receiving waters will probably be attained if plant effluents are limited to 20 ppb of phenol or phenolic equivalents and 15 ppm of five-day BOD and 15 ppm of suspended solids.

Some of the industries producing phenolic wastes are: coke, synthetic resin, oil refining, petroleum cracking, tar, road oil, creosoting, wood

distillation, and dye manufacturing plants.

(2) Chemical Wastes, Other than Phenolic

Adequate protection should be provided if:

(a) The pH of these waters following initial dilution is not less than 6.7 nor more than 8.5. This quality in the receiving waters will probably be attained if plant effluents are adjusted to a pH value within the range of 5.5 to 10.6.

(b) The iron content of these waters following initial dilution does not exceed 0.3 ppm. This quality in the receiving waters will probably be attained if plant effluents are limited to 17 ppm of iron in terms of Fe.

(c) The odour-producing substances in the effluent are reduced to a point that following initial dilution with these waters the mixture does not have a threshold odour number in excess of 4 due to such added material.

(d) Unnatural colour and turbidity of the wastes are reduced to a point that these waters will not be offensive in appearance or otherwise unattractive for the aforementioned uses.

(e) Oil and floating solids are reduced to a point such that they will not create fire hazards, coat hulls of water craft, injure fish or wildlife or their habitat, or will adversely affect public or private recreational development or other legitimate shore line developments, or uses. Protection should be provided for these waters if plant effluents or storm water discharges from premises do not contain oils, as determined by extraction in excess of 15 ppm, or a

sufficient amount to create more than a faint irridescence.

Some of the industries producing chemical wastes other than phenolic are: oil wells and petroleum refineries, gasoline filling stations and bulk stations, styrene co-polymer, synthetic pharmaceutical, synthetic fibre, iron and steel, alkali chemical, rubber fabricating, dye manufacturing, and acid manufacturing plants.

(3) Highly Toxic Wastes

Adequate protection should be provided for these waters if materials highly toxic to human, fish, aquatic, or wildlife are eliminated.

Some of the industries producing highly toxic wastes are: metal plating and finishing plants, discharging cyanides, chromium or other toxic wastes; chemical and pharmaceutical plants and coke ovens. Wastes containing toxic concentrations of free halogens and wastes containing resin and fatty acid soaps are included in this category.

(4) Deoxygenating Wastes

Adequate protection of these waters should result if sufficient treatment is provided for the substantial removal of solids, bacteria, chemical constituents and other substances capable of reducing the dissolved oxygen content of these waters unreasonably. In addition to sewage some of the industries producing these wastes are: tanneries, glue and gelatine plants, alcohol, including breweries and distilleries, wool scouring, textile, pulp and paper, food processing plants such as meat packing and dairy plants, corn products, beet

sugar, fish processing and dehydration plants.

GLOSSARY

MPN.....	Most Probable Number
ml.....	Millilitre
BOD.....	Biochemical Oxygen Demand
ppm.....	Parts Per Million
ppb.....	Parts Per Billion
pH.....	Hydrogen Ion Concentration
Fe.....	Iron

OBJECTIVES FOR INDUSTRIAL WASTES CONTROL IN ONTARIO

The Commission requests all industries discharging wastes to public waters to undertake effluent improvement programs to meet the following objectives:

Protection of Receiving Water

In keeping with The Objectives for Water Quality Control in the Province of Ontario and the overall policy of protecting water quality while recognizing essential use for wastewater disposal, the Commission may require the industry or industries contributing wastes not specified in the following, to limit, destroy, remove or modify any waste constituents that may be in question. This may apply to waste constituents that are not readily removed by conventional treatment and are only reduced by dilution and other natural stream purification processes.

In order to maintain acceptable water quality conditions in the receiving water, it may be necessary for the industry to provide more intensive waste treatment as the density of industrial and other development increases.

Acceptable Receiving Water and Effluent Characteristics

1. Biochemical Oxygen Demand (BOD) - The concentration of BOD at any point in the receiving water after initial dilution shall not exceed 4 parts per million by weight, and the concentration of dissolved oxygen in the receiving water shall not fall below 4 parts per million by weight, at any time.

2. **Suspended Solids** - The concentration of suspended solids in wastewater at the point of discharge to a receiving water shall not exceed the concentration of suspended solids in the industrial water supply by more than 15 parts per million by weight.
3. **Oils and Greases** - The concentration of oils and greases of vegetable, animal or mineral origin in wastewater shall not exceed 15 parts per million by weight at the point of discharge to the receiving water.
4. **Toxic Substances** - Materials or waste components that are toxic to aquatic life or render the water unsuitable for potable or recreational uses shall be eliminated or destroyed.
5. **pH** - The pH or relative acidity or basicity of wastewaters shall be controlled and maintained within the range of 5.5 to 10.6 in the pH scale.
6. **Taste and Odour** - Waste materials or components that impart tastes and odours to the receiving waters or to fish, that would render the waters or fish unsuitable for use, shall be eliminated or destroyed. Phenol or phenolic equivalents should not exceed an average of 2 parts per billion or a maximum of 5 parts per billion in the receiving waters following initial dilution.
7. **Aesthetic Qualities** - Treatment or control shall be effected to ensure that waste discharges do not impair the aesthetic qualities of the receiving water by imparting colour, by giving rise to accumulations of solids, oils or greases, by inducing foaming, or by other adverse effects.

A P P E N D I X C

BACKGROUND DATA

The following information was gathered by the Ontario Water Resources Commission at control stations believed to be representative of uncontaminated waters in the Serpent River Basin in the Elliot Lake area:

Chemical

Solids (Total)	- 43 ppm
Hardness	- 19 ppm as CaCO_3
pH	- 6.9
Sulphates	- 7 ppm
Nitrates	- less than .10 ppm
Iron as Fe	- less than .11 ppm
Manganese as Mn	- less than .25 ppm

Radiological

Ra-226	- .3 pCi/litre
U-238	- less than 1.0 ug/litre
Gross alpha	- less than 1.0 pCi/litre
Gross beta	- 9 pCi/litre

A P P E N D I X D

TAILINGS AREA - STRUCTURAL DETAIL

Research conducted by the field staff on the Stanrock tailings area during the summer of 1969 revealed that the suppositions presented in the 1968 report concerning the tailings area were essentially correct. Since the 1968 report was issued, much new information has been gathered and it will be presented in the form of a major study report. The report will cover all aspects of sulphide tailings disposal and will be extended to cover all sulphide operations in the Province of Ontario. The recommendations made in the report will be practical, not theoretical, and will serve at least to minimize the problem of mine waste drainage. Based on work conducted at Stanrock Uranium Mines Limited during the 1969 field season, the field staff feels that there is a slim but very real chance that a break-through, with regard to mine waste drainage, will be made in the upcoming field season.

The lowering of the decant weir at Stanrock Uranium Mines Limited forced the mass oxidation of the tailings within a radius of 300 feet of the decant pond. At the same time, the lowering of the pond level permitted the stream feeding the pond to cut 5 feet into the tails and as a result for the first time a limited but complete cross section of the tailings mass could be seen.

As predicted in the 1968 report, the intensity of the oxidation reaction is highly dependent upon the permeability and relative mineral composition of the beds of tails that were laid down as a result of tailings deposition within the basin.

Some of the beds at Stanrock are sandy with a low silt content and, at the time of deposition, were formed relatively near the source of material or in a relatively fast-moving stretch of the slurry stream. On the other hand, beds composed predominantly of silt sized particles were deposited in stagnant areas of the tailings basin or, at least, in an area remote from the source material. Since the slurry stream feeding the tailings area was moved occasionally, and because the slurry stream itself meandered aimlessly over the relatively flat surface of the tailings area and continually changed its course, the beds of silt and sand are found all over the tailings body, and are in no particular continuous, vertical or horizontal sequence. The relatively sandy beds tend to have a more open texture and hence, permit the percolation of water (and oxygen) throughout them. Favourable conditions for the survival of specific types of iron oxidizing bacteria are therefore set up. The beds composed predominantly of silt sized material tend to restrict the free passage of water (and oxygen). Furthermore, concentrations of clay minerals (which swell on contact with water) in the silty beds close the structure even more. Therefore, the production of a ferruginous cap (stable red coloured covering) on the surface of the tailings area is much more dependent on whether a silt bed or a sand bed lies at the surface at any particular location, than on the iron content of the beds. A silt layer lying on the surface of a tailings area will eventually, but not readily, form a stable iron oxide coating.

In the Stanrock tailings area, the tails in the vicinity of the decant

pond are completely unaltered (dark blue grey) at a depth of 3 feet 10 inches. Above this depth, the relative clastic composition of the beds, and the differential oxidation, migration and deposition of the contained iron compounds render the structure visible to the human eye.

Photograph A3

In the above photograph, the unaltered tails (in the bottom of the cut) are clearly visible. They are overlain by a highly oxidized layer that slopes gently to the surface. This bed is overlain by a relatively flat and relatively unoxidized layer of tails which in turn is overlain by a thin ($\frac{1}{2}$ inch thick) surface bed of highly oxidized tails (in this case, forming a ferruginous cap). The decant pond is in the background.

Photograph A4

Photograph A4 shows a dome of relatively unaltered tailings reaching the surface of the tailings area. The result therefore is the absence of a ferruginous cap in the area where the dome reaches the surface. Oxidized layers lap the edge of the dome. Where the oxidized layers break the surface, a ferruginous cap results.

Photograph A5

Photograph A5 shows the same feature as Photograph A4 but in greater detail. Notice the area with and without the ferruginous cap and the reason for both. Notice also that the massive oxidized bed lapping the limb of the dome is itself riddled with thin relatively unoxidized layers.

If a thin bed of silt lies on the surface of a section of the tailings area, it will tend to dry in the sun and break into polygonal sections. Water (carrying oxygen) is then able to more effectively gain entrance into the mass, the result being a ferruginous cap of inferior quality but of amazingly intricate design.

In the Elliot Lake district, the large size of the tailings areas results in relatively thin bedding structures in the contained tailings masses.

If, for any reason, a stable ferruginous cap is required over any particular tailings area or section of tailings area, it can be induced simply by ploughing the area with normal farm equipment to a depth of 6 inches to 1 foot. The overall effect is to break up (and therefore increase the surface area of) the

silty surface layers, and to expose underlying easily oxidizable layers to the correct atmospheric conditions.

A P P E N D I X E

IRON: Ferrous, Ferric

(in the Stanrock effluents)

(a) Source

As indicated in the GEOLOGY-MINERALOGY Section of this report, the composition of the ore conglomerates of the Quirke Lake Trough has been estimated to be as follows:

Quartz Pebbles	-	40 - 65%
Sulphides	-	2 - 8%
Sericite	-	5 - 20%
Matrix Quartz	-	15 - 25%

Pyrite (FeS_2) is the main sulphide mineral present. Pyrrhotite (Fe_{1-x}S) ($x = 0 \rightarrow 0.2$) is less abundant while other sulphides are rare.

The pyrite is present as euhedral to subhedral grains and in aggregates; the individual masses are usually between 14 and 100 mesh in size.

The present Stanrock main tailings area contains in excess of 7,000,000 tons of tailings grading approximately .7% iron as Fe. In other words, the main tailings area contains approximately 49,000 tons of iron -- generally in combination with sulphur.

Our gross loadings indicate that, at present, 2 tons of iron as Fe are being lost from the tailings area per day. Assuming steady state conditions, the iron content of the tailings area will be completely discharged in another 24,500 days (i.e., the year 2037).

(b) Reaction Chemistry

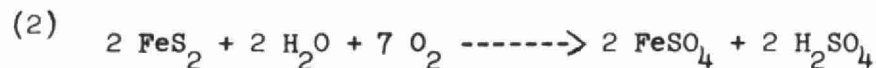
(i) Sulphide to Sulphate

When natural sulphuritic material in the form of sulphide, and usually in combination with iron, is exposed to the atmosphere (oxygen), it may theoretically oxidize in two ways with water (or water vapour) as the limiting condition. In the first circumstance, assuming the process takes place in a dry environment, an equal amount of sulphur dioxide will be generated with the formation of iron (ferrous) sulphate, thus:



Iron Sulphide + Oxygen -----> Ferrous Sulphate + Sulphur Dioxide

If, however, the oxidative process proceeds in the presence of a sufficient quantity of water (or water vapour) then the direct formation of sulphuric acid as well as iron (ferrous) sulphate in equal parts may be anticipated, thus:

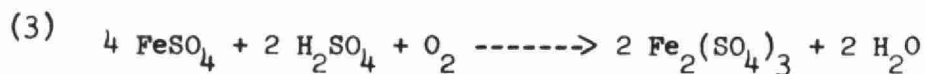


Iron Sulphide + Water + Oxygen -----> Ferrous Sulphate + Sulphuric Acid

NOTE: It should be pointed out that in the great preponderance of mine environments, there is strong indication that the latter reaction is favoured.

(ii) Oxidation of Iron (Ferrous to Ferric)

Iron (ferrous) sulphate in the presence of a sufficient quantity of sulphuric acid and oxygen oxidizes to the ferric state to form iron (ferric) sulphate.



Ferrous Sulphate + Sulphuric Acid + Oxygen -----> Ferric Sulphate + Water

Here water is not limiting since it is not a requirement for the reaction but rather is a product of the reaction.

The evidence seems to be rather conclusive that a specific bacterium is generally involved in the above equation, and is at least responsible for accelerating the oxidation of ferrous iron to the ferric state.

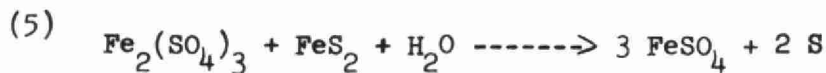
(iii) Precipitation of Iron

The iron (ferric) associated with the sulphate, then commonly combines with the hydroxyl ion of water to form iron (ferric) hydroxide. In an acid environment, iron (ferric) hydroxide is largely insoluble and accordingly, precipitates.

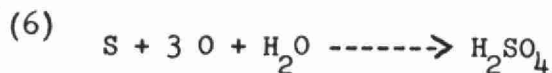


Ferric Sulphate + Water -----> Ferric Hydroxide + Sulphuric Acid

At the same time, it should be pointed out that another possible reaction has been suggested in the case of iron (ferric) sulphate. From an energy standpoint, the iron (ferric) ion may enter into an oxidation-reduction reaction with iron sulphide whereby the iron (ferric) ion "back triggers" the oxidation of further amounts of sulphuritic materials (iron sulphide) to the sulphate form, thereby accelerating the acid-forming process.



Ferric Sulphate + Iron Sulphide + Water -----> Ferrous Sulphate + Sulphur



Sulphur + Oxygen + Water -----> Sulphuric Acid

The fact that very little "free" sulphuric acid is found in mine waste drainage is probably due to the reactions between other soluble mineral species and H_2SO_4 .

FERROUS IRON

Equilibrium Composition for the System $Fe^{+2} - Fe(OH)_2$ at Various pH Values,
Expressed as a Fraction of the Total Fe^{+2} Present

pH	5.0	6.0	6.5	7.0	7.5	8.0	9.0
Fe^{+2}	.999	.903	.485	.087	.0094	.00095	.00001

FERRIC IRON

Equilibrium Composition for the System $Fe^{+3} - Fe(OH)_3$ at Various pH Values,
Expressed as a Fraction of the Total Fe^{+3} Present

pH	1.00	1.25	1.50	1.75	2.00	3.00
Fe^{+3}	.8677	.1947	.0979	.0213	.0085	.000008

The overall solubility of Fe^{+2} and Fe^{+3} in mine water is determined by many factors, none of which can be evaluated as an isolated variable. However, for both Fe^{+2} and Fe^{+3} , the limiting solubility at any pH is the solubility satisfying the equations:

$$K_{\text{sol}} \text{Fe}(\text{OH})_2 = \left[\text{Fe}^{+2} \right] \left[\text{OH}^- \right]^2 \quad \text{and}$$

$$K_{\text{sol}} \text{Fe}(\text{OH})_3 = \left[\text{Fe}^{+3} \right] \left[\text{OH}^- \right]^3$$

i.e.

<u>pH</u>	<u>Fe⁺², ppm</u>	<u>Fe⁺³, ppm</u>
1		56,000
2		560
3		0.56
4		0.00056
5		
6	29,400	
7	294	
8	2.94	
9	0.0294	

In the Elliot Lake district, the mine waste drainage effluents tend to change colour on standing while exposed to the atmosphere. Initially clear and colourless, the waste assumes a faint yellow flush which, in time, gradually deepens to intense amber-red.

The Initial Colour Change

The deep red colour is caused by finely divided suspended particles of hydrated ferric oxide. In other words, the ferrous iron in part oxidizes to the desirable ferric state.

The following rule of thumb is applicable in the Elliot Lake district:

When comparing from a chemical standpoint seepage flows, etc., "the worse the waste looks, the better it is". In other words, the pools of deep amber-red coloured waste that are frequently seen in the Elliot Lake area are chemically

more suitable for discharge to a watercourse than are the clear colourless wastes that discharge from the interior of the tailings areas. The amber coloured waste is generally lower in iron concentration and contains noticeably lower concentrations of Cu, Zn, Co, etc., than the raw seepage wastes. This fact in itself suggests that all mines in the Elliot Lake district that work the pebble conglomerate should, as a matter of policy, retain the effluent (decant and seepage) from their tailings areas as long as possible in an area downstream from the tailings area.

A P P E N D I X F

MICROBIOLOGICAL LEACHING OF SULPHIDE MINERALS

Microbiological leaching involves the oxidation of the sulphide portion of various metallic sulphides with the subsequent release of the metal values into solution. Three names have been given to bacteria capable of oxidizing sulphide minerals:

- (1) *Thiobacillus ferrooxidans*;
- (2) *Ferrobacillus ferrooxidans*; and
- (3) *Thiobacillus sulfooxidans*.

However, it has been shown that there is no justification for separating the bacteria into different genera and species and, as such, the name *T. ferrooxidans* has been suggested for the group.

Like many bacteria, *T. ferrooxidans* is ubiquitous, and its activities are manifested in leaching wherever a sulphide substrate, oxygen, carbon dioxide, water, certain essential nutrients and the correct pH make up a suitable environment.

T. ferrooxidans belongs to a rather select group of bacteria which are rather independent in temperament, and which probably existed on this earth long before those other microbial forms which are inter-dependent for many of their foods and conditions of life. In contrast to most bacteria, *T. ferrooxidans* uses atmospheric carbon dioxide as its sole source of the carbon necessary for the generation of cellular material. Rather than utilizing organic matter such as fats, carbohydrates or proteins as a source of energy, this organism cannot even tolerate their presence, and obtains its energy solely by the oxidation of inor-

ganic materials such as ferrous iron, or sulphur in the form of elemental sulphur or as metallic sulphides.

T. ferrooxidans has adapted itself to live and grow in the strongly acidic environments (pH 1.5 - 3.0) which result from the oxidation of sulphur, and in the presence of many heavy metals which are released into solution from minerals concurrent with the oxidation of ferrous iron and sulphides. In order for *T. ferrooxidans* to function in the biological leaching of sulphides, the pH must be below 4 and preferably below 3. Apparently the organism can oxidize sulphur at pH's as high as 5, but it will only occasionally oxidize sulphide at pH's above 3.5, and never above 4.0.

In attacking the sulphide moiety in crystalline or amorphous substances, the bacteria convert the sulphide to sulphate, and with the disruption of the solid matrix, the metal ions go into solution. These metals remain in solution as the water-soluble sulphates initially, although iron may be subsequently precipitated as the insoluble hydroxide or basic sulphate. This hydrolysis of ferric sulphate produces sulphuric acid along with either the ferric hydroxide or basic ferric sulphate. The sulphuric acid modifies the pH in the micro-environment immediately surrounding the bacteria. If there is no external disruption, the pH frequently tends to stabilize near pH 2 due to the resolubilization of ferric hydroxide. If there is excess pyrite or pyrrhotite present, it will go lower; pH values as low as 0.9 having been attained. pH values below 1.2 have a definite detrimental effect on the bacteria, interfering with their activity and resulting

in the production of elongated cells.

Although T. ferrooxidans survives in conditions that are highly toxic to most other forms of life, it still must have oxygen to live. Every pound of sulphur (either as native sulphur or as sulphide) requires two pounds of oxygen for complete conversion to sulphate. Similarly, the bacteria require 0.14 pounds of oxygen for every pound of iron converted from the ferrous to the ferric form.

Phosphate and ammonia appear to be the most critical nutrients with regard to the nutrient requirement of T. ferrooxidans.

Temperature is another factor which can influence biological leaching. The optimum temperature for biological leaching has been found to be 35°C. The bacteria are inhibited at temperatures of 40°C and above. When no other factors have become rate controlling, the rate of leaching decreases as the temperature decreases; as yet, however, no minimum temperature has been established. Leaching is known to occur slowly at 3 to 6°C.

Substrates Oxidized by Thiobacillus Ferrooxidans

<u>Substrate</u>		<u>End Product</u>
Ferrous Iron	Fe^{++}	Fe^{+++}
Trithionate	$\text{S}_3 \text{O}_6^{=}$	$\text{SO}_4^{=}$
Tetrathionate	$\text{S}_4 \text{O}_6^{=}$	$\text{SO}_4^{=}$
Thiosulphate	$\text{S}_2 \text{O}_3^{=}$	$\text{SO}_4^{=}$
Sulphur	S^0	$\text{SO}_4^{=}$
Sulphide	$\text{S}^{=}$	$\text{SO}_4^{=}$

Metallic Sulphides Oxidized by Thiobacillus Ferrooxidans

Arsenopyrite - $\text{Fe}_2\text{As}_2\text{S}_2$

Bornite - Cu_5FeS_4

Bravoite - $(\text{Ni}, \text{Fe})\text{S}_2$

Chalcocite - Cu_2S

Chalcopyrite - CuFeS_2

Cobaltite - CoAsS

Covellite - CuS

Enargite - $\text{Cu}_3(\text{As}, \text{Sb})\text{S}_4$

Marcasite - FeS_2

Marmatite - $(\text{Zn}, \text{Fe})\text{S}$

Millerite - NiS

Molybdenite - MoS_2

Orpiment - As_2S_3

Pyrite - FeS_2

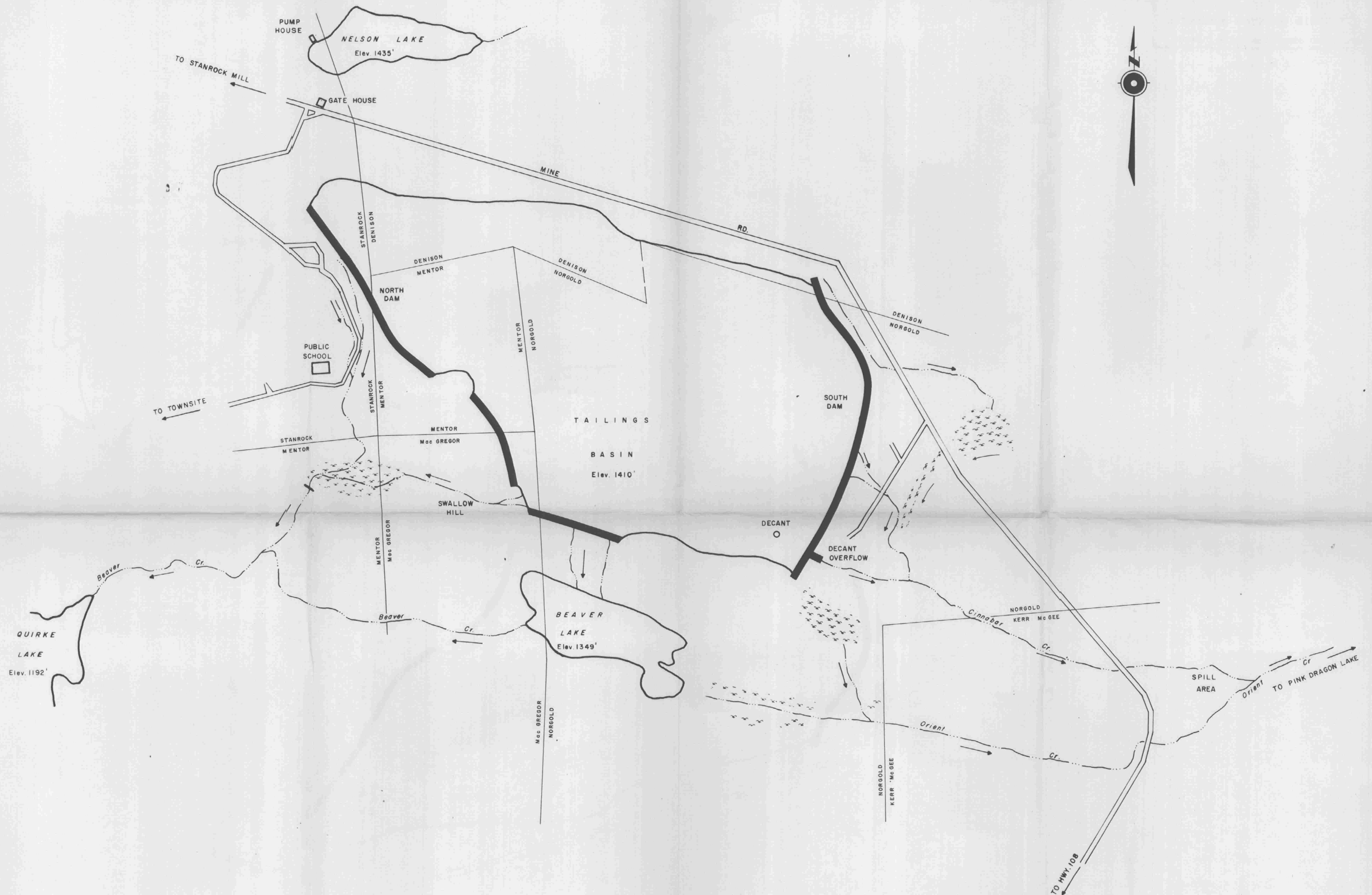
Pyrrhotite - Fe_7S_8

Sphalerite - ZnS

Stannite - $\text{Cu}_2\text{FeSnS}_4$

Tetrahedrite - $\text{Cu}_8\text{Sb}_2\text{S}_7$

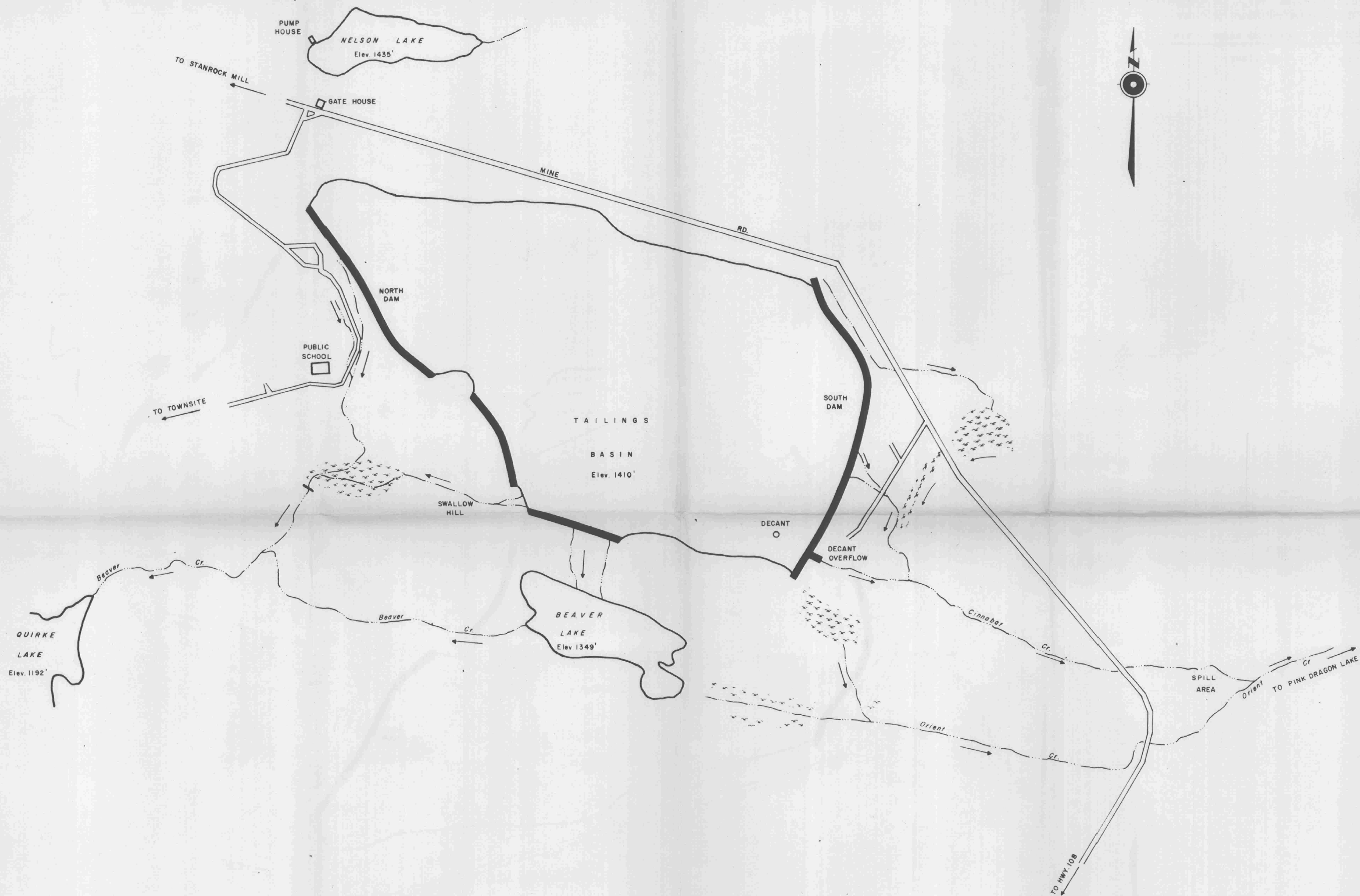
Violarite - $(\text{Ni}, \text{Fe})_3\text{S}_4$



SHEET No. 2

ONTARIO WATER RESOURCES COMMISSION
STANROCK URANIUM MINES LTD.
ELLIOT LAKE ONTARIO
CLAIM BOUNDARIES (TAILINGS AREA)

SCALE: 1" = 400'	
DRAWN BY: L. L. BROOME	DATE: DECEMBER, 1969
CHECKED BY: R. H.	DRAWING No: 69-226-1W



SHEET No. 1

ONTARIO WATER RESOURCES COMMISSION	
STANROCK URANIUM MINES LTD.	
ELLIOT LAKE ONTARIO	
TAILINGS AREA DETAIL	
SCALE: 1" = 400'	
DRAWN BY: L. L. BROOME	DATE: DECEMBER, 1969
CHECKED BY: R. H.	DRAWING No: 69-226-1W



LEGEND:
- TAILINGS DISPOSAL AREA
- SURFACE WATERS INFLUENCED BY MILL WASTES
- SAMPLING POINTS

SHEET No. 3

ONTARIO WATER RESOURCES COMMISSION
STANROCK AREA
ELLIOT LAKE ONTARIO
LAKES SYSTEM

SCALE: 0 1/2 1 2 MILES
DRAWN BY: S.G.H. & L.L.B. DATE: DECEMBER 1969
CHECKED BY: R.H. DRAWING No: 69-227-1W

RIO ALGOM
QUIRKE MINE
(Operating)

DENISON MINES
LTD.
(Operating)

PANEL
MINE
(Closed)

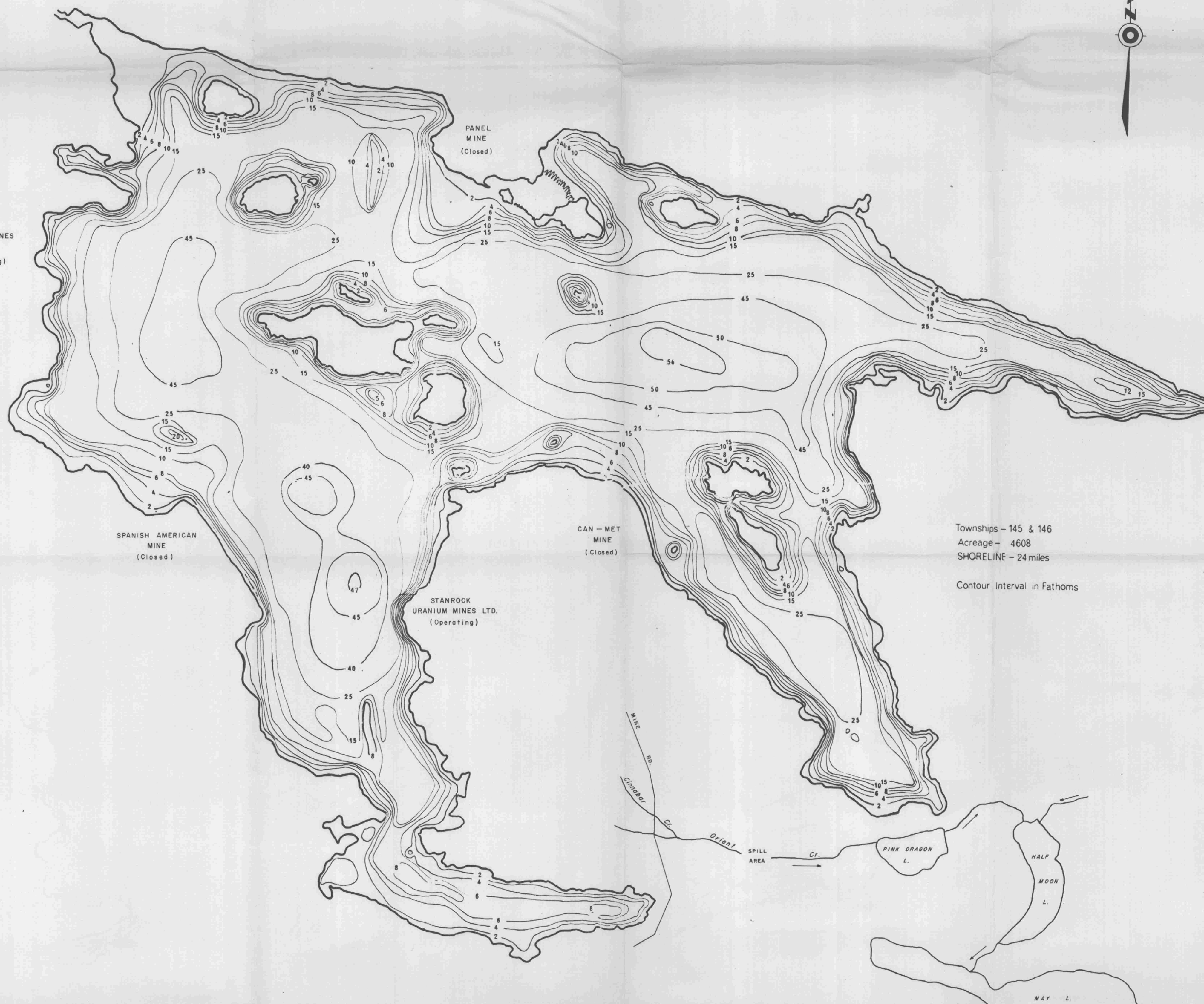
SPANISH AMERICAN
MINE
(Closed)

CAN - MET
MINE
(Closed)

STANROCK
URANIUM MINES LTD.
(Operating)

Townships - 145 & 146
Acreage - 4608
SHORELINE - 24 miles

Contour Interval in Fathoms



SHEET No. 4

ONTARIO WATER RESOURCES COMMISSION

STANROCK URANIUM MINES LTD.
ELLIOT LAKE ONTARIO
QUIRKE LAKE MINING AREA

SCALE: 1" = 1320'

DRAWN BY: L.L.B.

DATE: JANUARY, 1970

CHECKED BY: J.H.

DRAWING No: 70-2-1W